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FOREWORD

The Coal Division of the Institute, organized in 1930, has by its achievements fully proved the vision and foresight of its founders. Volumes containing the papers prepared for the Coal Division were published in 1931, 1932 and 1934. The present volume includes papers presented at the meetings held at State College, Pa., in October, 1934; St. Louis, Mo., October, 1935, and New York, February, 1935 and 1936.

The work of the Coal Division not only has been of incalculable advantage to the more progressive men within the industry, but, in addition, the class of technical papers presented at the various meetings has added to the scope and catholicity of the Institute. Coal mining, at one time looked upon as the Cinderella of the mining industry, now occupies a position comparable in engineering technique with the more complex branches of metal mining.

This condition has been brought about by a rapid translation from hand labor to mechanization. A growing number of coal properties now employ electrically driven coal-cutting machines, drills, coal-loading machines, and electric locomotives, this equipment supplanting the historic pick and shovel with the monotonous and wearisome work of less happier days. The growing installation of, at times, rather complex coal-cleaning methods, which have brought about a marked increase in the value of the product, has further expanded the necessity for engineering services, the increased investment required widening the gap between the former badly equipped mines and those of more modern construction.

Much of the surplus investment in excess mines and man power within the industry has been liquidated and multiple shifts are now being worked in many properties, which in turn has served to increase the annual output per dollar of investment. The combination of added machinery and increase in hours worked per week has also reduced the number of accidents per million man-hours worked, and a higher percentage of extraction has been made possible.

During the past two years, far-reaching attempts were made through the medium of Federal regulation to regiment and control the industry, first through the National Recovery Act and later by the so-called "Guffey" bill, the last enacted specifically for the purpose of controlling the Bituminous Coal Industry. Both laws were held by the United States Supreme Court to be unconstitutional. Throughout the tenuous life of these two laws, the Bituminous Coal Industry was hopelessly divided as to their merits and workability. Many thinking coal operators held the opinion that with competent, impartial fact finding, carried on by the United States Bureau of Mines, a nonpolitical branch of the government, the industry would make more progress toward stabilization than under Federal laws foundationed on labor-political pressure. Others favored the laws.

With mining machinery now well perfected and wholly past the experimental stage, the remaining and most fertile field for endeavor rests in the direction of improving employer and employee relations, a task now much simplified by the fact that it is no longer necessary to bring thousands of non-English-speaking men of foreign birth into the industry, this class of labor bringing with it the aloofness and too often the bitterness of spirit engendered by generations of poorly remunerated toil and class repression. Today, the high schools in the various coal-mining districts are annually turning out many thousands of young men, the sons of mine workers, all of whom are forward-looking Americans, who prefer the modern mining methods to those so tenaciously held to by their fathers. When this newer and better situation is fully comprehended by all employers, the strife and bickering that has marked the conduct of the coal industry will disappear. Men, whatever their station in life may be, will invariably respond to fair and considerate treatment.

EUGENE MCAULIFFE,
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Bumps in Coal Mines—Theories of Causes and Suggested Means of Prevention or of Minimizing Effects*

By George S. Rice,† Member A.I.M.E.

(New York Meeting, February, 1935)

The subject of violent bumps in coal mines has been again brought to attention by a recent succession of such occurrences in the coal mines of the Cumberland field of eastern Kentucky and southern Virginia, which have caused fatalities and injuries to underground employees, a report on which is being issued by the United States Bureau of Mines[1]. The A.I.M.E. has had two valuable papers on bumps in the No. 2 Mine, Springhill, Nova Scotia, each followed by general discussion on bumps and their causes[2].

This paper gives a brief review of suggested causes and remedial measures given in English, Canadian and United States reports on bumps in coal mines, and then presents suggestions arising from the writer's recent investigation of the bumps in the Cumberland mines, made at the request of the mine operators, to determine the causes and remedies. These bumps, which occur under the simplest natural conditions, provide better opportunity for analysis of causes and effects than afforded under more complex conditions. The simple conditions in this field consist of relatively thin and level coal beds and uniform roof strata free from folding and faulting.

DEFINITION OF BUMPS

Bumps in coal mines, as defined in this paper, are sudden violent bursts of coal that usually occur without warning in the extraction of pillars and may or may not be accompanied incidentally by falls of draw slate, upheaval of bottom and smashing of timbers. They are not known to occur where the advancing longwall method of mining, accompanied by pack-wall building, is employed, as it is so generally in Europe and a few places in the United States. This may be because the conditions necessary for producing them are lacking. The mining conditions where bumps occur usually prohibit the adoption of the advancing longwall method because of the difficulty and cost of obtaining sufficient pack-wall material for proper stowing of the excavations. In many instances it would have to be brought in from the surface, as practiced in some mines in Europe.

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* Published by permission of the Director, U. S. Bureau of Mines. Manuscript received at the office of the Institute Dec. 1, 1934.

† Chief Mining Engineer, U. S. Bureau of Mines, Washington, D. C.; Chairman, Committee on Ground Movement and Subsidence, A.I.M.E.

[1] G. S. Rice: Bumps in Coal Mines of the Cumberland Field, Kentucky and Virginia—Causes and Remedy. U. S. Bur. Mines *R. I.* 3267 (1934).

[2] W. Herd: Bumps in No. 2 Mine, Springhill, Nova Scotia. *Trans.* A.I.M.E. (1930) **88**, Coal Division, 151–191.

T. L. McCall: Further Notes on Bumps in No. 2 Mine, Springhill, Nova Scotia. *Trans.* A.I.M.E. (1934) **108**, 41–60.

In development work by headings bumps rarely occur. An example of one of the rare occurrences happened in a Utah mine where more or less parallel major faults made a vertical rock wedge in the roof; the slippage downward of the wedge caused a bump.

The term "bump" should not be confused with what is sometimes termed "bumping" in ordinary pillar extraction when the "roof is weighting" at the break line and heavy falls are impending, as after withdrawal of posts. Also, a bump is not like a general squeeze caused by insufficient pillars, which may crush coal, upheave bottom, and break timbers; it is a single burst that occurs usually without giving sufficient premonitory warning to the men working in close proximity, so they may escape death or injury from flying coal or debris. An air blast sometimes results from a large bump that throws out masses of coal or causes extensive falls of draw slate or "weak top."

In relation to the mechanics of bumps, the author has put forward in several previous reports and discussions the hypothesis that there are two kinds of bumps, which are quite distinct as to their mechanics and effects: (1) pressure bumps and (2) shock bumps.

1. *Pressure Bump.*—This type of bump occurs when the pillar is overloaded and bursts, just as a block of structurally strong coal or other material in a pressure-testing machine suddenly bursts when its bearing strength is exceeded. A pressure bump is equivalent to a rock burst in deep metal mines, such as occur in Michigan and South Africa.

2. *Shock Bump.*—This type occurs where a strong massive stratum, either immediately over the coal or higher up if not too far above, ruptures as a beam or flat arch and a ground wave is imparted to an already highly loaded pillar support. The author's investigations indicate that this is the principal type of bump that occurs in coal mines. Such bumps have occurred at the Coal Creek mines in the Crow's Nest Pass coal field of British Columbia, the Carbonado mine in Washington state, the Springhill mine in Nova Scotia, and in a dozen or so mines which in recent years have experienced bumps in the Cumberland field of Kentucky and Virginia.

After observing the result of certain effects after heavy shock bumps in several of the Cumberland mines, described later, the author has postulated that when the immediate roof is strong and elastic like a dense sandstone it not only springs down and back to its former position but may also be set in vibration under certain conditions of an elastic roof layer and crushed pillars.

CONDITIONS WHERE BUMPS OCCUR

Bumps in coal mines fortunately occur in only a few coal fields of the world, which have certain requisite geologic and structural features, as follows:

Natural Conditions

1. A depth of cover, generally more than 1000 ft., over the coal bed although this may be less where there are contiguous mountain masses.
2. A strong massive stratum of rock like sandstone or conglomerate in the strata above the coal bed being mined, provided that between the coal and the strong stratum there is not a structurally weak material such as shale, of considerable thickness, say not more than 5 to 10 times the thickness of the coal bed, which will cushion the effects of rupture of the strong stratum.
3. A strong coal. That is to say, coal that does not crush in a pillar under a unit load of 4000 or 5000 lb. per sq. in. when the width of the pillar is more than three or four times its height.
4. A strong floor at least as strong as the coal; that is, not a thick soft underclay.

Mining Methods

A combination of the foregoing natural conditions is essential to the causing of bumps. The mining conditions that bring this combination into play are as follows:

1. Taking out too much coal in development work so that the static load[3] on the remaining coal in pillars is greatly increased. For example, if the cover were 2000 ft. deep and 50 per cent of the coal were taken out in advance, the static unit load on the pillars would be about 4000 lb. per sq. in. This is assuming that, with an extensive area of excavation, the support from the sides is negligible. With most coals this unit load of 4000 lb. would be close to the crushing strength of small pillars of coal. If, on the other hand, only 25 per cent of the coal were taken out on the advance under a cover 2000 ft. thick the unit static load would be about 2600 lb. per sq. in. which would be well within the bearing strength of most coals.
2. An irregular pillar withdrawal line, leaving narrow pillar points projecting into a goave, may receive a load so close to its bearing strength that a small impact produced by a rupture of the rocks above might cause a violent bump.

---

[3] The term "static load" as used here means the load before ground movement begins, which may throw an additional load on the pillar or pillars.

As indicated above, it is not alone the method of mining that causes the bump phenomenon; there must be the combination of the natural conditions listed. Even bad mining methods do not produce violent bumps where these natural conditions do not prevail. The coal-bearing sediments in most coal fields of the world are predominantly shale with more or less interbedded shaly sandstones. Even where there is a thick, strong sandstone or conglomerate stratum in the overburden, if there is considerable thickness of weak rocks, like shales or thin-bedded sandstones, between this and the coal bed pieces and blocks spall off from the roof and more or less loosely fill the arch or dome formed over the goaf[4].

Examples of Conditions Where Bumps Have Occurred

South Staffordshire, England.—In mining the so-called 10-yd. seam or "thick coal" of South Staffordshire, "bumps" have been experienced at intervals for many decades, and it is thought that the term "bump" originated in that coal field. In bump occurrences there the coal bursts from the face and sides of roadways without warning. Sometimes small volumes of inflammable gas are released, timbers are broken or thrown out of position, and the floor may be heaved. Fatalities and serious injuries of persons in the immediate vicinity have resulted.

The natural conditions are that the coal bed is 18 to 30 ft. thick, and where bumps have occurred it lies at depths of 1500 to 1800 ft. The roof above the bed has sandstone layers either directly above the coal or higher. The coal is structurally strong and the floor is strong. The coal bed dips are usually less than 4 or 5 degrees.

The method of mining used in most of the mines of the field is unique; square or rectangular chambers are excavated, leaving a surrounding pillar 30 ft. thick and within each chamber six to eight small coal pillars 24 ft. square are formed by roadways driven in the bottom coal and then widened to 30 ft. Subsequently the coal above each roadway is taken down by picking and barring down layer by layer. By this method of mining about 62 per cent of the coal is taken out ultimately but during this extraction bumps occur sometimes without warning.

From the bumps effects noted by the writer when visiting some of the mines of this field and from study of the discussions on this subject in various British publications, these bumps generally appear to have resulted from overloading of the pillars and may be "pressure bumps."

The static unit load on the coal before mining development is about 2000 lb. per sq. in., and if the extraction is 62 per cent there would be a static unit pressure of about 5000 lb. per sq. in. on the pillars when formed

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[4] In this paper the British term "goaf," rather than "gob," which is more generally used in America, is employed as more expressive of the meaning of worked-out territory, which may or may not be more or less filled with gob.

Considering that the pillars are only as wide as they are high, it is not surprising that they yield under pressure.

In certain mines in South Staffordshire modifications of the pillar-and-stall method of panels have been tried, but this method has not avoided the occurrence of bumps.

A committee of the South Staffordshire and Warwickshire Institute of Mining Engineers was appointed in 1930 to work in conjunction with a Safety in Mines Research Board investigator to obtain information looking to better control of the hazards.

The Committee rendered its first progress report in January, 1933[5]. Although the final conclusions have not been reached, these preliminary suggestions of the committee, which are here briefly abstracted, on the causes of bumps in the South Staffordshire "thick coal seam," are of great interest:

> It is obvious that the working of a thick seam without proper packing must result in irregular settlement and rupture of large masses of the strata.
>
> . . . the strength of a pillar of coal to withstand crush will vary with the thickness and structure of the seam and the crushing strength of the coal. It is also obvious that the well defined cleat of the coal will have the effect of decreasing the strength of the pillar . . .
>
> The sandstone roof, which is very strong and compact, does not break down until a large area of coal has been extracted. During this period it forms an arch which extends back into the waste but which is supported on a buttress along the face . . .

Coal Creek Mines, Crow's Nest Pass Coal Field, British Columbia.—The coal in these mines is of Cretaceous age and is a coking coal. The beds lie below a high, mountainous plateau. Beginning in 1907, a series of bumps occurred in these mines, sometimes accompanied by outbursts of gas, and causing some fatalities and injuries, culminating in one of great magnitude which occurred Nov. 8, 1916, which produced earth tremors felt for many miles in the surrounding country. Following this, the writer was asked to investigate the cause of such bumps and the relation to instantaneous outbursts of gas, a phenomenon which also occurred in this field. His report[6] suggested that the bumps originated in the rupture of a thick conglomerate stratum, which lay 1000 ft., more or less, above the coal bed that was being mined, and this rupture and the resultant fall of a great mass of rock through a small space made by subsidence of more flexible strata below it imparted a shock wave down through these strata to the roof of the coal bed.

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[5] *Trans.* Inst. Min. Engrs. (1933) **85**, pt. 3, 116–137; followed by a discussion, 137–147.

[6] G. S. Rice: Bumps and Outbursts of Gas in the Mines of the Crow's Nest Pass Coalfield. Brit. Columbia Dept. Mines *Bull.* 2 (1918). The investigation found that the gas outbursts were phenomena separate from the bumps except incidentally in some cases.

The immediate roof was a weak shale, and an impure top coal, which was thrown down into the workings over an area of nearly 40 acres, caused a wind blast. The total thickness of overburden is 2000 to 2500 ft. where the more severe bumps have occurred in the coal field. The remedy proposed by the writer was to take out not more than 15 per cent of the coal in advance development and to retreat from boundaries with long straight break lines. This plan was adopted in part and appears to have been successful in preventing very severe bumps but not lesser ones.

State of Washington.—Bumps causing fatalities occurred some years ago in the Black Diamond coal mine, Washington, in a bed of noncoking coal of Eocene age, 42 to 66 in. thick, which lies in a gently folded anticline, the axis of which dips 15° to 20°. The mine was developed by the usual room-and-pillar method in pitching seams and no trouble was experienced until a depth of cover of 1400 to 1600 ft. was reached, and then only when the pillars were being drawn at one side of the anticlinal arch. The effect of this extraction, through rupture of the arch, was to send shocks upon already overburdened pillars acting as supporting piers or abutments. The cover consisted predominantly of sandstone layers and it appeared that the arch strata, extending over the workings, had a wide span before it ruptured.

The effects observed by the writer after one of the bumps were very peculiar apart from the bursting of the coal ribs. In certain workings at or near where the bump had struck, open spaces were left over the coal several inches high, which extended laterally 5 or 6 ft. in from the side of the pillar. Also, an open crib of round logs, which had been on one side of an entry, had been moved to the other side without displacement of the logs that composed it. This indicated that the roof was more or less elastic and after compressing the coal along the edge of the pillar, the roof had sprung back and possibly vibrated. The primary cause of the bumps evidently resulted from the irregular manner of withdrawing the pillars in a critical area receiving the thrust of anticlinal arches, so that the remaining portions of the pillars were stressed beyond their bearing strength. Subsequently, when greater care was exercised to withdraw pillars systematically, serious bumps did not occur, so far as the writer is informed.

Carbonado Mine.—This mine, in Washington, was troubled with bumps some years ago, which caused accidents. The mine produces a coking coal of Eocene age, the beds are 48 to 60 in. thick and dip about 60° at the outcrop, flattening to 20° with depth. Where the bumps occurred the cover was about 2000 ft. The sandstones and sandy shales predominate in the roof measures. The immediate roof is strong; also the floor.

The room-and-pillar system was employed, the rooms driven to the rise, and the mined coal run down chutes to cars on the haulageway below. When the rooms reached the level above, the pillars were drawn back so that several rooms were being driven up the rise and at the same time pillars were being drawn back down the pitch by side slicing. The remaining pillars or parts of pillars were thus subjected to heavy loading and the unit loading on any particular pillar was increased by its irregular shape and method of withdrawal which proceeded down the pitch. The latter way of withdrawal may have caused more weighting on the upper end of the pillar.

On several occasions bumps threw off coal from the ribs violently, killing or injuring the men in the immediate vicinity. The remedy recommended and adopted was to have more careful laying out of room pillars and to withdraw them in groups in a more or less straight line of break. Also, a plan was proposed of having the main haulage levels driven within the footwall, as frequently employed in steep-pitching metallic-ore veins. This method has since been more or less adopted in some steeply pitching Pennsylvania anthracite mines.

Nova Scotia.—In 1924, following a series of bumps in the Springhill No. 2 mine, Nova Scotia, which had been increasingly serious and affected the morale of the workers, the writer was invited by the Nova Scotia Department of Public Works and Mines to examine the conditions and if possible to suggest such measures as in his opinion would lead to a remedy and cessation of these dreaded accidents. His findings were published by the Department in December, 1924[7] and these are further detailed in a paper by the late Walter Herd, Chief Mining Engineer of the British Empire Steel Corporation[8]. Only a brief statement will be given here of the features receiving consideration.

The No. 2 coal bed in which the bumps occurred is of Carboniferous age and the coal is an excellent steam and domestic coal. At the outcrop the bed dips about 30°, flattening to 20° with depth. It averages about 9 ft. thick, is free from impurity bands and is a somewhat blocky coal. The floor is strong shale. The immediate roof is also a strong shale about 15 to 30 ft. thick. Above this shale massive sandstone lenses are present in parts of the mine where bumps are experienced. The presence of these was found by vertical drill holes, which showed them to be 20 to 50 ft. thick, or more, in places. It is the sudden breaking of this massive sandstone, acting as a beam or arch under heavy overburden due to depth, that is considered to be the cause of the bumps.

The method of mining in the upper part was a bord-and-pillar system adapted to a steep inclination but when the workings reached a depth of 1600 ft. of cover, serious caving of the working places occurred. Then the size of pillars was increased and the width of bords was decreased to 10 ft. As the workings went deeper, further changes were made, but when the depth of cover approached 2000 ft. bumps began occurring with increasing frequency and severity with depth.

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[7] G. S. Rice: Occurrence of Bumps in the Springhill No. 2 Mine, 1924. Nova Scotia Department of Public Works and Mines (1924).

[8] Reference of footnote 2.

In the conclusions in the report of the writer in 1924, he suggested in effect that if natural and economic conditions permitted the employment of the advancing longwall method with pack walls that would be the best method to adopt, but as there was insufficient material available for pack walls, except by the prohibitively expensive method of bringing it in from the surface, he proposed retreating longwall with faces parallel to the dip.

This plan was heartily endorsed by the company and adopted in principle, as described in the paper by Herd, previously cited. Unfortunately bumps continued to occur. This the writer considers was due, in part at least, to carrying on the retreat on too narrow a front and contiguous to the old goaf on the rise side. Mr. Herd, however, concluded that:

> Experience with bumps for a considerable number of years leads the writer to the opinion that under certain conditions of overlying and underlying strata no known method of mining will eliminate them and allow of coal production on a commercial basis, but it is often possible to establish conditions which may nullify or at least mitigate their effects.

One of the remarkable and significant bumps described by Mr. Herd is that of April 12, 1928. (See pages 167–171, previous citation.) This bump occurred where the cover was 2500 ft. deep. It had struck on the interior of a large coal pillar along a narrow strip which paralleled the strike of the bed and was 33 ft. from a haulage level on the rise side of the pillar. The strip began 215 ft. back from the retreating face and extended for 90 ft. The coal was crushed down along this strip and left a cavity above it 10 ft. wide and, at the center, 18 in. deep, as found later when the pillar was mined out. The roof over the cavity, Herd comments, was "intact." The floor below the crushed coal was buckled up as a ridge into it; the solid coal, 33 ft. wide, between this and the level was moved en masse and uncrushed up the rise, so that it protruded several feet into the level.

Mr. Herd thought that the crushing of the coal was due to buckling up of the floor and that after the movement of the solid coal into the level the crushed coal dropped down, leaving the cavity above. The writer of this paper, however, suggests that the buckling with wedging of the solid coal and the crushing were simultaneous effects, and further suggests the possibility that the crushing of the coal in the strip might have been due to the thrusting down of the roof by a shock wave from above and the springing back of the roof towards its original position.

In either case an inspection of Herd's Fig. 6 shows that the face was retreating parallel with an old goave 300 ft. distant up the pitch and torsional stresses, which might have been produced in the strong sandstone stratum overlying the bump area, may have caused its rupture.

The writer, in the discussion that followed Mr. Herd's presentation of his paper, suggested that the method of mining adopted—retreating longwall—"as yet had not been sufficiently advanced to determine its success or failure because it was too near the old workings."

T. L. McCall, who succeeded Mr. Herd as Chief Mining Engineer of the company operating the Springhill mine, described[9] further bumps that had occurred and presented the results of tests that he had made on convergence of roof and floor, using the methods developed by the Safety in Mines Research Board of Great Britain. His graphs indicate that convergence began more than 400 ft. from the retreating face and at central measuring stations opposite the middle of the face the compression of the coal was as much as 4 to 6 in. when the face was 250 ft. away and then the compression increased rapidly until it was 26 in. when the face was about 30 ft. away. During these tests bumps in different places in the general area occurred within 250 ft. of the face, and after each bump the compression indicated at stations in the vicinity increased more rapidly.

It appears to the writer that the respective areas were under pressures approaching the maximum bearing strength of the coal when it would begin to flow and that any rupture in the strong roof stratum would be likely to cause a bump. He concurs with Herd and McCall in the difficulties of preventing bumps under the conditions but suggests that although the retreating face had been lengthened to nearly 500 ft., it was adjacent to an extensive goaf on the rise side and any cantilevering of the sandstone stratum over the goaf might throw an additional load on the coal adjacent to the face. Moreover, the next step in the retreat is about 600 ft. behind the first, which would tend to prevent regular breakage of massive sandstone. It is only when cantilevers of such a stratum can be broken regularly that relief will be given to the excessive pressure on the coal adjacent to the retreating faces.

Cumberland Coal Field of Kentucky and Virginia.—The series of bumps that have been occurring during the last few years in many coal mines of the Cumberland field give unusual opportunity for the study of bumps under the least complicated surrounding physical conditions; viz., level coal beds of about the same hardness and same thickness, 4 to 5 ft., similar strong floor and strong main roof strata, the latter containing one or more massive sandstone layers, and also freedom from faults and local folds.

It is true that the thickness of overburden at the different mines varies greatly, ranging from nothing at the outcrop to 2500 ft. under the higher mountain ridges but this variation is an advantage in studying the effect of greater thickness of overburden on the increasing liability of bumps where there is no complication of steep dips of the beds and presence of faults and rolls.

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[9] Reference of footnote 2.

The main cleats or vertical faces of the coal are well developed and approximately parallel the axis of the coal basin, which bears about northeast-southwest. There are butt cleats at right angles to these, but less regular.

The basin is about 90 miles long and 15 to 20 miles wide, bounded along the northwest by the Pine Mountain ridge, formed by an overthrust fault, and on the southeast by the Cumberland Mountain anticline. The beds at the sides of the synclinal basin are steeply inclined but the mines in which the bumps occur are in high erosion-formed ridges in the middle portion of the basin, and in these the measures are level and now relieved from the effects of geologic thrusts.

The coals are in the Pottsville measures of the Carboniferous age, and are high-grade gas and byproduct coking coals. There are about eight beds of workable thickness in one place or another but in any one place usually only two are of workable thickness under present market conditions.

The workable beds are very similar in character and usually are free from impurity bands. The floors of these beds are either hard shale or sandy shale. The immediate roof over the respective coal beds varies in different places from a strong shale to a sandstone, but the main roof generally, if not always, carries a strong massive sandstone at varying distance above the coal. Judging from outcrops and prospect drill-hole reports, over 70 per cent of the coal-measure strata is either sandstone or sandy shale.

The mountain ridges have steeply sloping sides, so that the load of the covering strata is received directly on mine pillars; that is, there is no lateral support as in the case of continuous strata as there may be in a level country. On the contrary, the higher sloping ground adjacent to that vertically above certain pillars may throw additional weight upon these pillars. This is indicated by the occurrence of bumps at a place in one mine where the height of cover directly above was only 500 ft., whereas most bumps in this field do not occur unless the cover is 1000 ft. or more.

Mining Methods in the Cumberland Mines

The plan of development in all the mines of this field is room and pillar, varied only in detail. Main entries enter the outcrop and are either double, treble or quadruple. The room entries are generally driven on the butts and the rooms at right angles to the coal faces. Only one mine was an exception, the rooms being driven quartering on the faces.

The greatest variations in the different mine layouts is in the width of rooms and width of pillars and methods of drawing the pillars. The majority of the present mines take out 60 to 70 per cent of the coal area in the advance work and now in pillar extraction retreat work are experiencing bumps under high cover, which are causing fatalities and injuries.

Faulty Mining Methods.—Faulty mining methods may be summarized as follows: (1) Taking out too much coal in the development; (2) pillars too small; (3) in pillar withdrawal, (*a*) not drawing back the pillars in a straight line; (*b*) narrowing the pillars at the retreat line by diagonal slicing of the pillars, thus making points projecting toward the goaf; (*c*) pulling the pillars in separate panels rather than by a long retreat line, extracting entry pillars as well as room pillars.

The general methods employed in the field are no worse than in most coal fields of the country, where roof strata are chiefly shales and where there is no difficulty in breaking the roof on withdrawal of timbers. There are a few mines in the field in which the development methods are excellent in respect to not taking out more than 25 per cent of the coal on the advance, but even in these mines the pillar extraction is faulty in not taking out entry and room pillars on the same extraction line. Where the cover is less than about 1000 ft. in these mines, with some exceptions, no trouble from bumps is experienced, but under higher cover serious bumps have occurred.

It should be emphasized that in this field, even where 60 per cent of the coal is extracted on the advance, bumps are experienced only in pillar drawing. The immediate roof is always strong and requires little or no timbering in entries; props in rooms are not withdrawn but some of them —about every third one in a line—are shot out to induce falls when a wide excavation has been made. However, the breaks do not go high until a very wide excavation is made.

Because of the freedom from impurity in the coal and general absence of draw slate, there is little waste material in the goaf. In some places the immediate roof was so strong that it was holding up over the goaf for spaces 100 or 200 ft. wide back from the pillar line. Even when falls are induced, usually they are not thick, and the breaks observed are sloping upward over the goaf, leaving lower layers cantilevering, the main roof evidently then acting either as a beam or flat arch buttressed against the pillars.

Details of Some Bumps.—The mines that have experienced bumps are described in detail in a Bureau of Mines publication (reference of footnote 1).

The severity of the individual bumps is not measured by fatalities or injuries, as many occur when no one is in the vicinity or in the mine. Measured by quantity of coal burst out, they have varied in this field from a few tons to 1500 tons of coal.

At one of the mines where two beds were worked, data were obtained as to the height to which bump effects would go under the particular conditions. A bump in the lower bed (Harlan seam) caused cracking in the strata reaching a mine in an upper bed (Darby seam) 220 ft. above. The cracks extended across the main entries in that mine and were tension cracks ½ in. wide. These did not appear until three or four weeks after the bump. The total cover here was 800 ft. thick.

In another case a bump struck a pillar which was not the nearest to the goaf but the second from it in a position that seemed protected. After the bump, which killed a man close by although four others nearer the goave escaped, even though surrounded by loose coal, it was found that the pillar bumped had a space or hollow extending to the floor. Many similar instances are reported of hollows made in pillars where bumps centered. These and other effects have led the writer to the conclusion that practically all bumps in the Cumberland field are "shock bumps," striking pillars already loaded to near their bearing strength.

Gas Conditions.—As the mines in this coal field are rated as nongassy and have exposed outcrops on one or several sides, little or no gas would be expected, but following two of the bumps ignition of firedamp occurred. In one case the body of gas was so great that its explosion killed a man. A test bore hole near a retreat line in one of the mines, which was cased and sealed, was found filled with gas. It seems evident from this that the pressure on the coal will release occluded gas; i.e., gas contained in the pores of the coal or absorbed in its mass.

Possible Vibration Effects from a Bump.—A most unique effect of a bump was experienced in one of great severity. The floor was buckled up in places but the sandstone roof was not. In this area there was a whitish clay band 1 to 2 in. thick over the coal, which filled shallow cavities in the immediate sandstone roof. The bump had burst out the coal from pillar ribs, which with other debris amounted to about 1500 tons to be cleaned up. The whitish clay was found thickly scattered over the debris in the entries, giving an appearance of rock-dusting with limestone dust. Wide open spaces 4 to 6 in. or more high were found between the more or less crushed pillar and the roof—much more space than had been occupied by the 1 or 2-in. band of the whitish clay.

The writer concluded that it was a case of a shock bump produced by rupture of a strong sandstone stratum some distance above the coal acting as a beam or flat arch. The maps indicated that its span was 200 to 300 ft. over the worked-out area between the pillars bumped and the pillars in an opposite retreating face.

CONDITIONS INCREASING LIABILITY OF BUMPS IN CUMBERLAND MINES

The writer, in his report on the Cumberland bumps, concludes that the likelihood of occurrence in Cumberland mines is increased when:

(1) The pillar or pillars being excavated project into the goaf. There appears to be particular danger in drawing back entry pillars with goaves on either side.

(2) When two retreat lines are drawing away from one another.

(3) When two retreat lines of opposing panels or blocks are passing by one another.

It is also thought that:

(4) In drawing away from a boundary pillar or a large area of solid coal there is a critical distance, which may be 200 to 400 ft., depending on the strength and distance above of the overlying rock strata, height of overburden and thickness of the immediate roof if this is shaly. If it is shaly it is likely to break down and more or less fill the goaf. This filling is favorable in preventing severe bumps.

Uncertain factors in the present state of knowledge are:

1. Whether the occurrence of bumps, other things being equal, is more likely to take place when the retreat line is parallel to the cleat of the coal rather than when it is at right angles or diagonally crossing the cleat.

2. Whether bumps are more likely to occur when the retreat is parallel to the vertical joint planes of the overlying rocks. It is not known whether these planes are coincident with or parallel to the cleat of the coal, which could only be determined by a careful geologic survey through the field or at least in the vicinity of the mines subject to bumps.

Recommendations for Preventing or Lessening Occurrence of Bumps in the Cumberland Mines

Tentatively the writer suggests that it might be better to have the retreat line cross the line of cleat of the coal at right angles or diagonally, but whether or not this proves to be the case, he regards it as highly important that in the retreat the chain pillars of entries be extracted on the same line as the room pillars and not left projecting into the goaf. The retreat line may be in steps, but they should be regular and short.

The most drastic recommendations the writer makes is to cushion the fall or bending of the roof in the goaf by regularly spaced rock-filled cribs[10]. He believes, from his inspection along the edges of the goaves, that with the strong but compressible support given by parallel lines of rock-filled cribs there would be likelihood for the immediate roof cushioned by the cribs to bend down gradually in the goaf, then that successive layers of strata above would bend or else break gently over the pillars. The tensile strength of rock is relatively low as compared with its great compressive strength; therefore in a rock stratum, acting as a

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[10] Crib: A rectangular frame of timber, usually round, laid horizontally like the walls of a log cabin. In Great Britain and in the Anthracite district of Pennsylvania this is called a cog, but the term "crib" is more generally used in bituminous mining districts of the United States.

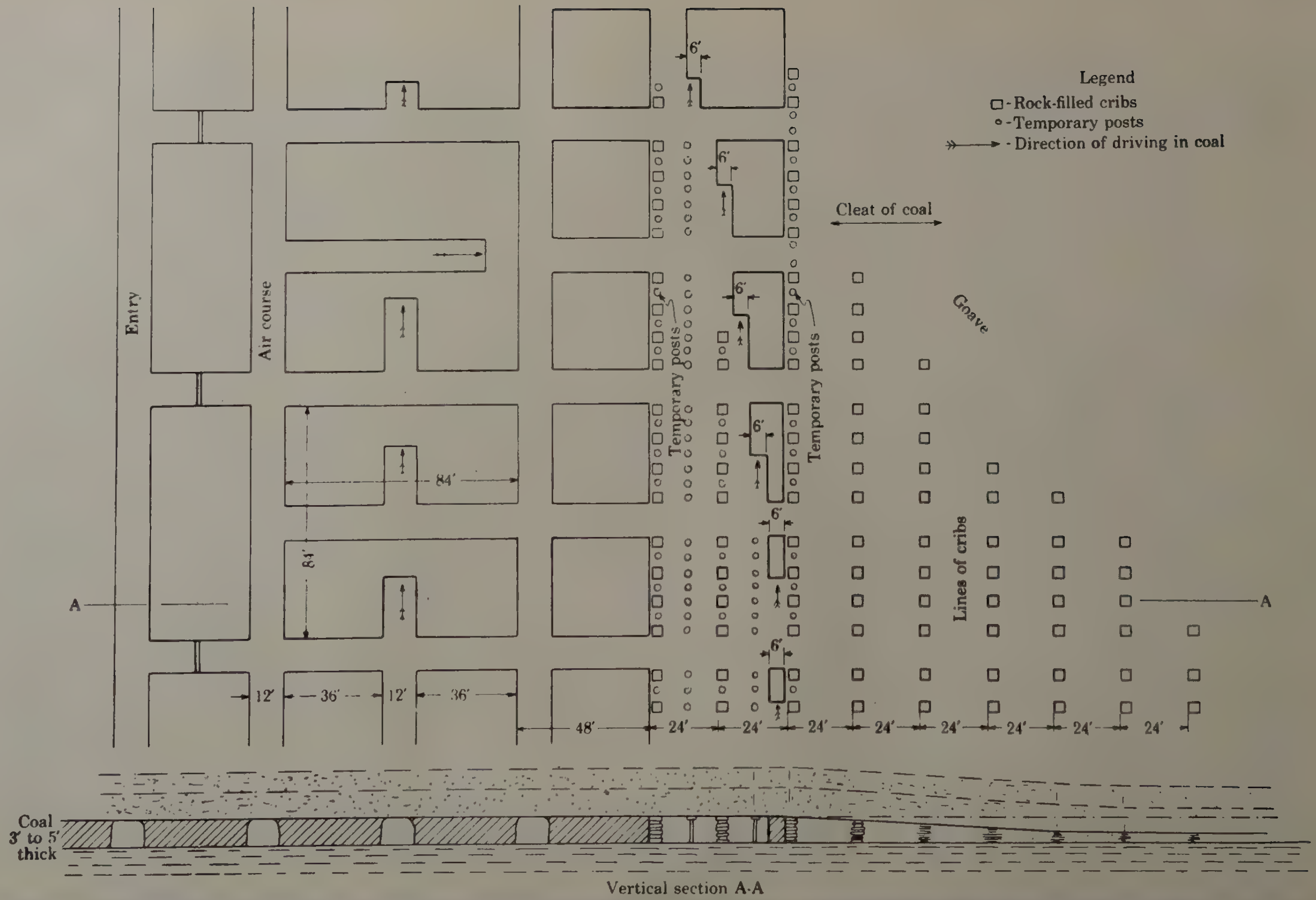

FIG. 1.—SUGGESTED METHOD OF USING ROCK-FILLED CRIBS AND STRAIGHT RETREAT LINE IN EXTRACTING PILLARS IN A UNIFORM COAL BED,

cantilever projecting out over the pillars, the break would be by tension in its upper part (as indicated in the vertical section in Fig. 1), an action which it is conjectured would not produce violent shocks.

With the present system the immediate roof is suddenly released in the goaf by collapse of the posts, and an opening is made above that practically equals the thickness of the coal bed. Then the successive layers above, acting as beams over the goaf, rupture suddenly by crushing of the rock in the central upper part of the beam (as indicated at *a-c* in Fig. 2) when the span becomes too long for the compressive strength of the rock. The result is to produce a violent shock, which is transmitted as a bump to the pillars acting as abutments.

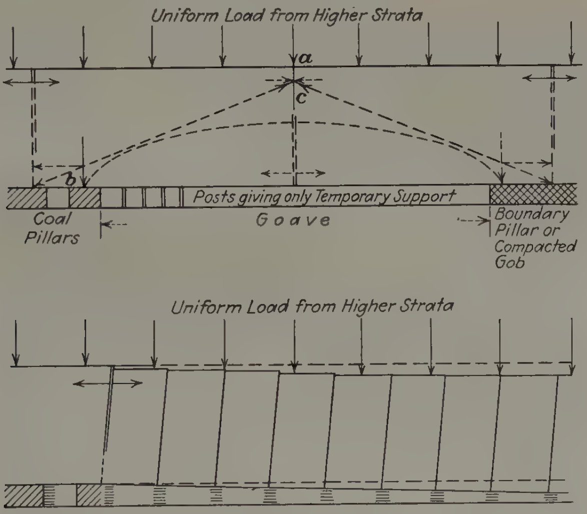
Uniform Load from Higher Strata
a
c
b
Posts giving only Temporary Support
Coal Pillars
Goave
Boundary Pillar or Compacted Gob
Uniform Load from Higher Strata

FIG. 2.—MODES OF RUPTURE OF A STRONG STRATUM OVER A COAL BED IN RELATION TO THE CAUSE OF BUMPS. ARROWS SHOW DIRECTION OF STRESSES.

I. Compression failure of rock between *a* and *c* may cause bump at *b*, whether stratum acts as continuous beam or arch.

II. Where stratum is elastically supported by rock-filled cribs in normal condition of beds it may, because of tensile weakness of rock, bend over pillars or break as shown.

Longwall Methods.—The advancing longwall method has been suggested as a remedy and it would probably be successful in preventing bumps if the excavations were systematically and completely packed, but from the writer's practical experience in longwall operations he does not believe it could be carried on with commercial success in the Cumberland mines under the mining conditions. These would require blasting large amounts of roof rock or else taking in material from the surface in order to build pack walls. The cost per ton of coal produced would be further increased by maintenance of the more permanent roadways with pack-wall sides.

Retreating longwall, or its equivalent, the extraction of large pillars on a straight line, and, where there is heavy overburden, the use of rock-filled cribs to cushion and allow the gradual subsidence of the roof, seems to the writer the most practicable plan for preventing severe and dangerous bumps.

Some of the mining men of the Cumberland region commented, when the writer informally proposed it, that this support would throw the weight of the roof back on the pillars. The writer does not think that this result would follow, but, on the other hand, under present practice the strong roof strata overhang far over the coal pillar retreat line, being practically without any support in the goaf except weak props, and this overhang must increase the normal unit load on the adjacent pillars.

STRENGTH OF CRIB AND ROOF SUPPORTS

In considering the question of the roof support by rock-filled cribs or cogs, attention is called to the full-scale testing of roof supports by the United States Bureau of Mines for the Pennsylvania Mine Cave Commission[11].

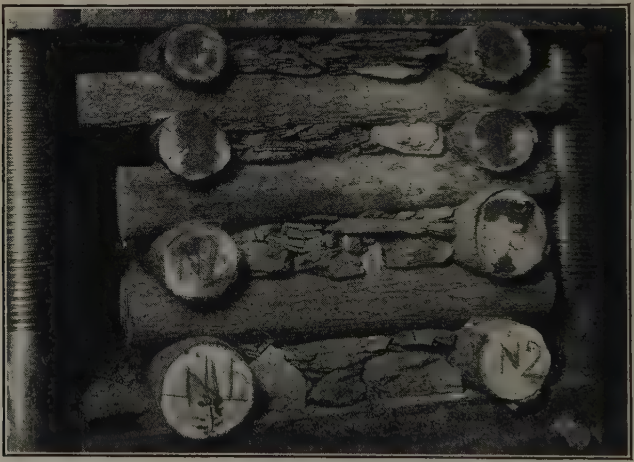

FIG. 3.—BEARING STRENGTH OF ROCK-FILLED CHESTNUT-OAK CRIB. Beginning of test by U. S. Bureau of Mines.

Test 9 (*Bull.* 303, p. 14) was made on a 5 by 3½-ft. rectangular cog (crib) 4.91 ft. high, built of 14 round, unbarked chestnut timbers 8 to 10 in. in diameter, filled with rock, ashes and sand (Fig. 3). The cog was subjected to successive increases in load and the compression measured for each increase in load. At 4,200,000 lb. (Fig. 4) the compression was 26.73 per cent. The maximum load applied was 6,400,000 lb., or 363,840 lb. per sq. ft. of area of the cog (Fig. 5) at which load the reduction in height was 46 per cent of the original height. An unfilled chestnut-oak cog of approximately the same size (Fig. 6) under a load of only 1,640,000 lb. (Fig. 7) compressed about 49 per cent. These tests demonstrated the enormous increase in strength by packing a cog with rocks in which fine material is interspersed.

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[11] Tests of Strength of Roof Supports Used in Anthracite Mines of Pennsylvania U. S. Bur. Mines *Bull.* 303 (1929).

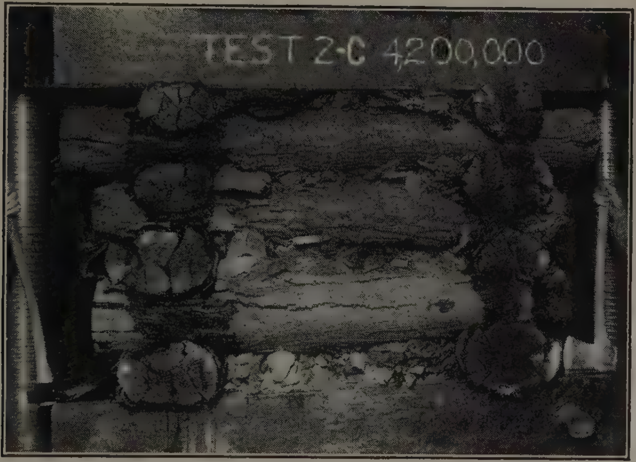

Fig. 4.—Compression of rock-filled chestnut-oak crib under load of 4,200,000 pounds.

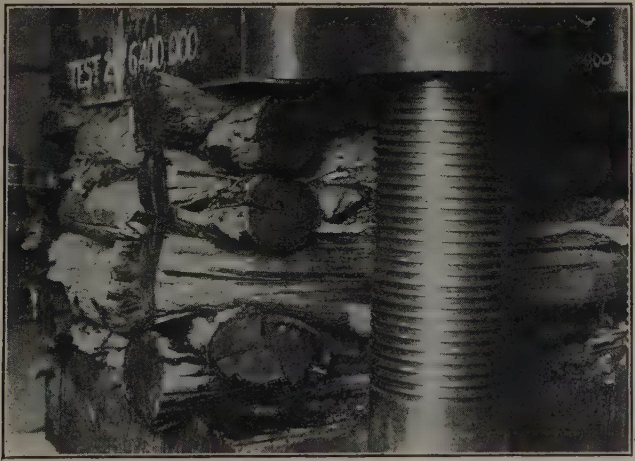

Fig. 5.—Compression of rock-filled chestnut-oak crib under load of 6,400,000 pounds.

As concerns the question whether the rock-filled cribs will have sufficient strength to provide any support for strata which unsupported will cause bumps, it is conjectured that the load will not be the entire weight of rock from the surface but only that of a limited height. For example, the case previously cited of a bump in one mine not having seriously affected a mine 220 ft. above it suggests that strata less than

200 ft. thick and possibly not more than 100 ft. thick would have to be given sufficient support to prevent violent ruptures and a consequent bump. The writer had previously postulated that the violent ruptures occur mainly from compression failure in the upper layer of such a strong stratum acting as a beam or flat arch. The thickest single massive sandstone in the coal-bearing strata, known of at any of the Cumberland mines visited, is about 60 ft. thick.

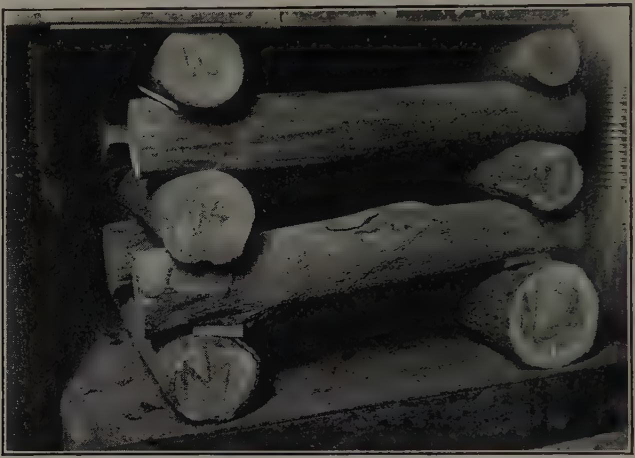

Fig. 6.—Bearing strength of chestnut-oak crib, unfilled. Beginning of test by U. S. Bureau of Mines.

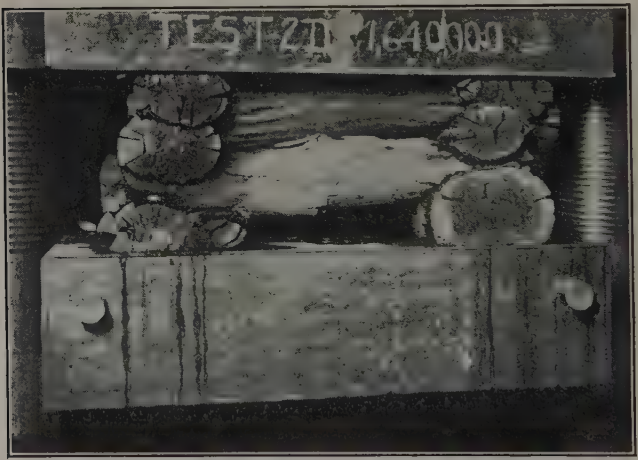

Fig. 7.—Compression of chestnut-oak crib, unfilled, under load of 1,640,000 pounds.

There is no indication from the reports of the specific bumps in the Cumberland mines that after a rupture has produced a bump, breaks of higher strata at the same place have produced noticeable bumps. It is probable that after the first rupture and fall of a broken stratum the space then made above that stratum would be gradually filled by spalling, bending and breaking of higher strata without transmitting a sufficient shock wave through the broken ground to cause another bump at the place of the first.

Under the plan shown in Fig. 1, such a crib would receive a load from an area of roof about 12 by 24 ft., or 288 sq. ft. If we assume that the top of the first strong stratum over the coal bed is 150 ft. above it, the weight of a column of rock of this height and area is about 2500 tons. The rock-filled chestnut-oak crib, cited above, supported 3200 tons when under a reduction in height of 46 per cent of the original height.

This does not mean that the crib or cog collapsed at that pressure, as there was no indication of collapse (Fig. 5). Increased loading probably would cause further compression until the rock material was compacted to practically the density of solid rock.

It is, of course, true that the cost of rock-filled cribs would be considerable unless more brushing were done in the main roadways, which would seem desirable from the operating standpoint in haulage and ventilation. Further, the temporary props now used, which are all lost in the goaf, could be recovered and re-used, which would offset some of the cost of the cribs. If filled cogs or cribs were used, their spacing and the cost per ton produced is conjectural and could be determined only by trial.

A suggested layout is indicated in Fig. 1, and it is believed it would be advisable from a safety standpoint to have this line diagonal or approximately at right angles to the cleat of the coal.

When conditions incident to the mine layout make it advisable, short steps or right-angle jogs in this extraction line might be made at intervals of 200 or 300 ft., provided these are not more than about 20 or 30 ft. long, to avoid too deep a re-entrant angle in the retreat line.

It will be noted in this plan that in the initial advance from the butt or room entries, the rooms and cross headings form pillars about 84 ft. square. Those close to the retreat lines are cut by headings into pillars about 36 ft. square. Meantime filled hardwood cribs, about 4 by 5 ft. in area and 8 ft. apart, are constructed in the butt headings, forming continuous parallel lines 20 ft. apart

The inby smaller pillars are to be mined off in successive slices, about 6 ft. wide, the width depending on the length of the cutter bars of undercutting machines if these are used. The slicing should begin from the inby crosscut and slices taken off successively until the respective pillar is completely extracted and the goaf reached. In taking the last slice the working place is protected by the previously placed line of cribs with temporary props between them, other lines of props to be installed as needed as the slicing proceeded but all props to be withdrawn when the pillar is extracted, leaving the cribs to support and cushion the roof in the goaf as it falls or bends downward.

It is thought that except under the highest cover and most severe conditions a smaller number of cribs than is shown could be used.

With the dimensions of pillars and the spacing in the figure, for a coal 4 ft. thick one crib would be required for about 48 tons of coal produced in the pillar extraction. Although the cost of the cribs would be a considerable item of expense in the cost of coal produced, there would be compensating advantages and savings in the method to offset the cost, besides the merit of lessened hazard to the miners.

SUMMARY

Coal-mine bumps are a comparatively rare phenomenon, experienced in few mining districts. This paper has reviewed briefly reports on districts in which they have occurred, which have been visited by the writer, in Great Britain, British Columbia, Nova Scotia and the United States.

The respective districts in the countries named in this paper are perhaps the principal ones in which coal bumps causing serious accidents have occurred. Technical journals have referred from time to time to bumps in coal mines of France, Germany, and other countries, but data of sufficient definiteness have not been available to the writer for inclusion of descriptions in this paper.

Only one natural condition of geologic formation, and an essential one, is common to all mines experiencing bumps; namely, the presence in the strata overlying the coal bed of a strong, massive rock stratum. Usually this is a thick, structurally strong sandstone but sometimes is a conglomerate. In all other respects, a wide variation in the conditions related to the occurrence of bumps exists in the different districts. These are:

1. Thickness of the coal as mined, ranging from 3 to 24 feet.
2. Dip of coal ranging from level to 20°.
3. Depth of cover, ranging from 1000 ft. (exceptionally as low as 500 ft.) to more than 2500 feet.
4. Character of surface topography from level to mountainous.
5. Methods of mining, although all the mines except one, employ room-and-pillar methods, the amount of advance extraction of coal varies from about 20 to 60 per cent, and the method of drawing pillars also varies widely.

It seems certain that there are two ways that bumps in coal mines occur:

1. By pressure of the overburden exceeding the strength of coal pillars, causing them to burst. This type of bump has been termed a "pressure bump."

2. By rupture of a strong stratum above, which it is postulated must be present for a bump to occur, and this rupture causes a violent shock wave to be transmitted like an earthquake shock to the pillars. This type of bump, which is the more usual one, has been termed a "shock bump."

Much discussion has followed previous papers as to whether the bumps result from rupture of the overlying strata as a beam or as an arch. The writer, in the light of his recent studies, believes that there is no essential difference in effect whether a continuous stratum ruptures either as a uniformly loaded arch or as a beam, which spans an opening. In either case it is the strength of the rock in compression at the middle of the span and in its upper part (probably one-fifth or less of its thickness), which determines when the arch or beam will fail with increasing span and by sudden failure transmit a shock wave to the pillar abutment at one end and to the boundary pillar, or a gob pile, at the other end of the span.

Remedial Measures must vary with the natural conditions. The most important are that:

1. The advance development should not take out more than 20 per cent of the coal area.

2. The retreat should be made by taking out all the pillars, both entry and room pillars, on straight and long retreat faces.

3. Lines of rock-filled cribs be placed adjacent to the retreat faces and the cribs be left in the goaf to cushion the downward movement of the roof and cause it to bend, if possible, or else to cause roof strata, which may cantilever from over the pillars, to break at the retreat face by tension in the upper part of each cantilever. Thus to avoid compressive rupture of the rock stratum, acting as a beam or arch, which gives a much more violent effect.

Although the foregoing remedial measures may not give immunity from bumps under all conditions, it is thought that the appropriate ones will go far toward lessening the liability of occurrence of bumps.

DISCUSSION

(*H. G. Moulton presiding*)

W. B. Plank,* Easton, Pa.—This is the first time I have heard such a detailed description of this phenomenon. I am interested in it from this standpoint: I wonder whether you have ever made any seismographic studies on the surface, which might possibly be correlated with the manifestations of bumps that you have observed underground.

If your theory that these bumps may be caused by successive failures of the rigid sandstone stratum above is correct, and I am rather inclined to think it is, these bumps ought to be preceded by some preliminary shock: in other words, this is an earthquake on a small scale. This would result in considerable movement of the earth's crust back and forth, and that, to my mind, would account for the terrific crushing of the coal. The roof moving back and forth above the pillar would certainly crush a considerable amount of the coal and throw it out into the open spaces; and if, as I suggest, this is an earthquake effect, it certainly would be preceded by minor shocks that could be detected, certainly, by seismographs on the surface.

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* Head of Mining Department, Lafayette College.

I am suggesting this as perhaps a means of detecting the time for occurrence of these things. This may be entirely impractical, but it occurs to me that seismographs can also locate the possible origin of these movements, these coming movements in the mine workings, and it may be possible to abandon the section of the mine affected until the stresses are neutralized.

G. S. Rice.—There has been, so far as I know, only one seismograph installed at a coal mine. I advised in 1917 the installation of a seismograph at the Coal Creek mine, British Columbia, following the great bump, but there were not many bumps after that. I have asked for a report, but have not received one; nevertheless I now doubt whether vibration precedes a crush rupture.

As far as the shock from a great bump is concerned, that was very evident at the Coal Creek mine, where the shock was felt for a distance of seven miles. But I do not think the shock is felt until after the rupture has occurred unless it is the culmination of a series of lesser breaks.

H. G. Moulton,* New York, N. Y.—A seismograph has been used in South Africa, I believe.

G. S. Rice.—Yes, to give warning of so-called "rock bursts," which are equivalent to bumps in coal mines, and also in the Kolar gold mines in India. But I do not think the South African Commissioners on Rock Bursts found the seismographic record taken on the surface of value in giving warning of major outbursts. There were too many and too complicated vibrations coming from lesser falls.

G. W. Evans,* Seattle, Wash.—It has been my pleasure to work with Mr. Rice on several of these problems and it has been my good fortune to study the conditions surrounding the bumps in the Crow's Nest Pass coal fields in southeastern British Columbia and at the Cassidy mine on Vancouver Island, as well as the bumps in the Black Diamond mine and the Carbonado mine in the State of Washington; also the bumps in the Sunnyside mine in the Utah coal fields and similar phenomena in the Cumberland mine, Wyoming.

I notice on page 17 the statement by Mr. Rice as follows: "Also, a plan was proposed of having the main haulage levels driven within the footwall, as frequently employed in steep-pitching metallic-ore veins. This method has since been more or less adopted in some steeply pitching Pennsylvania anthracite mines."

Mr. Rice is probably referring to the rock gangways and rock return airways developed within the Miller seam in the Carbonado mine. The bumps in the Carbonado mine were in the Wingate seam and did not develop until the mine was under cover of very nearly 2000 ft. The Wingate seam has an unusually strong roof and a very strong floor and it also has strong sandstone strata some distance above the coal bed itself. The bumps in the Wingate seam were caused by the robbing of pillars at distances of approximately 2000 ft. and, no doubt, were brought about by the crushing of the remaining pillars when the crushing strengths of such pillars were reached.

The driving of rock gangways and rock return airways in the rock underlying the coal bed was inaugurated in so far as the State of Washington is concerned in the Miller seam at Carbonado, but this was not done with the view of overcoming bumps and bumping conditions, but rather to be relieved of excessive amounts of retimbering in the gangways in the Miller seam. The Miller coal bed overlies the Wingate at a distance of approximately 300 ft. and has a great deal of carbonaceous matter in both the floor and the roof. When distances of 600 or 800 ft. were reached in the Miller seam, excessive squeezes and heaving took place in the main haulageway and return airway. Driving the gangways in the underlying rock and the return airways in the underlying rock, then working the coal bed with a stepped longwall system, overcame the difficulty of squeezing, but the cost was rather high.

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* Consulting Mining Engineer.

I used this same system of driving rock gangway at the Beaverhill mine, in the Coos Bay district, Oregon, in 1919 to overcome squeezing conditions in that mine. Also, I used a steep longwall system of mining the coal and materially reduced the cost of maintenance and the production of coal.

I believe that rock gangways driven in the anthracite fields of Pennsylvania are more for the purpose of reducing excessive maintenance costs in the haulageways than to relieve bump difficulties. I have in mind outlining a rock gangway in one of the collieries in Shamokin field, but this was to avoid excessive repair work and not to be relieved of bumps and bumping difficulties.

The subject of bumps is important enough for the Institute to appoint a special committee headed by that dean of mining engineers in subjects of this kind, Mr. George S. Rice, and have this committee make a further study of bumps and bring the information down to date. Coal mines throughout the United States and Canada are becoming deeper and the question of bumps will be an ever-present one.

L. HOLLAND,* Birmingham, England (written discussion).—Having had nearly 20 years experience in working the thick coal of South Staffordshire, where bumps are of frequent occurrence, I am particularly interested in this subject.

The most serious bumps occurred at Hamstead colliery, where the coal was 22 ft. thick and contained two bands of dirt, one of which was from 9 to 18 in. thick and the other varied from 1 to 4 ft. in thickness. Both bands were of a soft nature. The depth of cover was about 1900 ft., and in addition to strong sandstone beds about 30 ft. above the seam, the upper measures contained great thickness of red sandstone above the coal measures.

Mr. Rice has outlined the method of work, but a little amplification is necessary. In opening out the pit, main roads 10 ft. wide were driven out to the boundaries at 90 ft. apart, and bumps occurred in these roads. After several years working, other main roads were driven at right angles to the original roads in one district and at a distance of about 1000 ft. from the nearest working. Very severe bumps causing fatalities occurred in these roads, which were at right angles to the cleat. Eventually a fire occurred in one of the roads, due to the crushing of the coal, and another pair of roads was driven, commencing 900 ft. nearer the shaft in order to recover the roads beyond the fire area. These roads thus formed the base of a large triangle, and were at an angle of nearly 45° with the cleat.

Bumps in these roads were quite as severe. In the worst cases the floor was lifted several feet, and the face of the heading was forced out, while the roof frequently was unaffected. Severe bumps rarely occurred inside the panels of work. The worst bumps occurred in driving the roads to form a new panel, just before the old panel was finished, which is different from Mr. Rice's experience, and may be due to different conditions, such as strength of the roof and thickness of strong rock. The opinion I formed was that the pillars of coal left in the panel held up the sandstone roof, the binds below the sandstone fell between the pillars, but gave no support to the sandstone owing to the thickness of coal removed. When sufficient weight came on the pillars they were crushed and the sandstone arched with one leg, over the solid coal in the area where the roads would be driven. When the sandstone roof reached the breaking point, it would break over the solid coal and the sudden movement of an enormous mass of strata, even though the actual movement was small, has the same effect as a miniature earthquake, and the only relief valve in the solid coal was the heading.

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* President, Institution of Mining Engineers, Great Britain.

The intensity of some of these bumps may be judged by the fact that residents in the neighborhood, not necessarily immediately over the workings, have felt the vibration and ornaments in their houses have moved.

At this colliery gas was rarely present after a bump, so that gas can be ruled out as a contributory cause.

The position of the roads in the seam, whether in the middle or near the floor, did not appear to make any difference with regard to bumps.

At Baggeridge colliery, the thick coal is worked back longwall in panels 300 ft. wide, the bottom 7 to 8 ft. of the seam is partly packed and the top coal is got between and over the packs, and the immediate roof about 10 ft. thick comes down soon after the coal is removed. There is not so much support to the roof as in the pillars in the panels at Hamstead. Bumps are not so frequent or severe as at Hamstead but they occur further outby from the workings than at the latter colliery. The conditions at the two collieries, however, are not quite comparable because the roof at Hamstead was stronger than at Baggeridge.

With regard to remedial measures, if the headings were driven much more slowly the bumps would not be so severe, but the costs of repairs would be much heavier, owing to the floor lifting and the roof breaking.

In the cases stated by Mr. Rice, the conditions of working are quite different, as a comparatively large percentage of coal is worked in the process of development and all the pillars are finally extracted, whereas in Staffordshire and Warwickshire in developing the thick coal as few roads as possible, and these never more than 12 ft. wide, are driven, on account of the liability to spontaneous combustion.

In these cases without knowing actual conditions I agree with Mr. Rice in principle that a smaller percentage of coal worked in the process of development would have a beneficial effect, because the more the natural support is removed the greater is the pressure on the pillars.

With regard to leaving permanent cribs in the goaf to support the roof, there is a possibility of too much support being given to the roof, thus throwing the weight on to the pillars. Experience would soon prove if this were the case. Much depends on the height of the strong rock above the coal and the thickness of it, and in my opinion if the rock could be broken down more frequently, i.e., in smaller masses, there would be less shock, and therefore less severe bumps.

At one colliery in South Staffordshire an attempt was made to work the thick coal in separate sections by longwall retreating and taking about 7 ft. of coal in each section, but it gave rise to so much spontaneous combustion and caused so much loss of coal that the method was abandoned. Bumps occurred in driving the roads to split up the panels.

In Warwickshire the thick coal is worked in three sections by longwall retreating, the faces being carried 30 ft. behind each other, the two meading faces being packed solid but the last face having little or no packing. Severe bumps do not occur in working this coal, owing probably to the fact that the rocks above the coal are not so strong as in the South Staffordshire coal field.

G. S. Rice.—The information given by Mr. Holland is extremely interesting and will be of value to those studying bumps. There is one comment, however, about which I feel some question; that is, if the cribs are too strong it may throw the weight on to the pillars. I cannot see how weight can be thrown by putting in more support. It would seem to me rather that the strong cribs might prevent breakage of the rock over the goaf, but in the case where I am proposing it I am hopeful that the rock will not break but will bend.

L. Holland (written discussion).—When I wrote my earlier discussion, I had not then received Mr. Rice's *Report of Investigations* on bumps.[12] This report enabled me to understand the method of working much more clearly. It also sets out the geological structure. After reading this, it was clear that the roof in the Cumberland coal field is much stronger than the roof of the South Staffordshire thick coal, and accounts for the temptation to remove so much coal in the advance work, which undoubtedly brings severe weight on the small pillars remaining. The working of the pillars in some cases is very irregular and reminds one of the working of the thick coal at Hamstead colliery in the early days when bumps were very frequent, as also were fires in consequence.

The conditions were bad when I took charge of the colliery in 1903, and the number of fires for that year was 78, and as the area was already laid out, it was only possible to reduce the severity of the weight to a limited extent, and reduce the number of fires by one-half. In 1916 we leased a new area and it was possible to lay it out on better lines from experience gained in the old area. All developing roads were driven 70 yards apart instead of 25 or 30 yards, and the panels were worked back in a regular line, and not more than three panels at a time equivalent to a length of 230 yards. These panels constituted a district, and when this district had been at work about 18 months, a sufficient time for the roof to break down and settle, another district was commended, and in a further 18 months another one, so that there were then three districts working in steps, each with solid coal on one side and solid goaf on the other side.

From the opening up of this area until I left in 1923, there were certainly plenty of minor bumps, but none of any magnitude, such as frequently occurred in the old area. There was not a single fatal accident, nor a very serious personal injury from this cause, and fires became a comparatively rare occurrence. The experience bears out the recommendations in Mr. Rice's Cumberland field report.

The roof conditions and the method of work affect a decision on the question of putting in cribs, and as the immediate roof in the Cumberland coal field is stronger than that over our thick coal, judging by the area that is being supported on props, it appears to be quite likely that the cribs would allow the roof to sink more steadily and evenly and would take some of the pressure off the pillars if they are worked back in a regular line.

I assume that the reason why the coal is not worked by longwall method is because the roof is so strong and there is a scarcity of packing material.

T. L. McCall,* Sydney, N. S. (written discussion).—The bumps in the Cumberland field, together with those in the state of Washington, are peculiar in respect of the seams being thinner than in the other cases cited.

I believe that Mr. Rice can be credited with originating the appropriate terms of "pressure bumps" and "shock bumps."

There are few mines in which bumps occur where it is possible to study in part the actions of the rocks overlying a seam subject to bumps, consequently it is of great interest to note on page 22 a reference to a bump in a lower bed causing cracks to appear in an upper bed—the interval between the two beds being 220 ft.—about three or four weeks after the bump. The value of this information would be enhanced if it were known whether during this interval of time there were any bumps or rock jarring in the lower bed such as might be expected by this delayed rupture of rocks. It would also be helpful to know the extent of coal extraction during the lapse of time and the relation of the cracks in the upper seam to the workings in the lower seam.

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[12] G. S. Rice: Bumps in Coal Mines of the Cumberland Field, Kentucky and Virginia—Causes and Remedy. U.S. Bur. Mines *R.I.* 3267.

* Chief Mining Engineer, Dominion Steel & Coal Corporation, Ltd.

In the succeeding paragraph Mr. Rice makes mention of another bump, the peculiarity of which was that it struck a pillar that was not nearest the goaf but second from it, from which data the author apparently concludes that this was an example of a shock bump. However, I question that conclusion, because the cause of this bump might easily have been pressure acting directly on the pillar that burst. No mention is made of the condition of the pillar intervening between the one affected and the waste. This is a point of very great importance because one of the conditions necessary to cause a pillar to bump is that the pillar must be capable of resisting heavy pressure, otherwise should there be no resistance in a pillar the coal in it will merely flow when pressure comes on.

In the Springhill mines referred to in Mr. Rice's paper, before the method of working was changed the coal was won by room and pillar retreating from the boundary. When the main level had reached its boundary, inclines were turned away on the full rise of the seam, and when these reached the waste of the next level above, three rooms were turned off at the top and on either side of the incline, leaving pillars about 35 ft. wide. Thus, off a headway one set of rooms driving outby would have nothing but solid coal outside of them while the other three rooms, once the district had been fully opened up, would be driving towards waste. The worst bumps almost invariably happened in the rooms driving towards the waste, but no bumps occurred in drawing the pillars, the reason being that all resistance had been crushed out of the pillars. Such was the extent of this crush on the pillars that after a bump in one of the rooms one could often see the lights in the next room over the top of the pillar. This fact confirms what so many observers have recorded; viz., that the roof is apparently elastic and bounds up again after a bump.

It is quite possible that similar conditions may have existed in the case cited by Mr. Rice and referred to above, and that the crushing of the pillar next to the waste having destroyed its powers of resistance the pressure may have been transferred to the next outby pillar, which as it finally bumped must have had considerable powers of resistance.

It is difficult and often rash to venture an opinion without having had an opportunity to study conditions in the field, but I am somewhat sceptical as to whether the plan proposed by Mr. Rice in Fig. 1 will be entirely successful. Anybody who has had experience with bumps and has read of the physical conditions of mining in the mines in Cumberland County as described in Mr. Rice's report[13] will recognize by looking at the plans contained in the report that the mining methods employed enormously increased the chances of occurrences of bumps. Mr. Rice's proposed plan of working eliminates many conditions now favorable to causing bumps, but unless in his plan the pillars in the process of being drawn next to the waste have had the resistance crushed out of them, I am afraid that these pillars will be subject to pressure bumps.

In Springhill there was a bad bump under a condition somewhat akin to this. When longwall mining was first introduced, great difficulty was experienced in keeping the faces open, and frequently a face closed in and had to be reopened by ribbing up a new face. In one particular instance a combination of circumstances—when the full face was regained—caused a block of coal to be left at the top end of the face between the new face and the waste. The miners were warned to leave this block alone but the coal in it worked freely and was too great a temptation, so they kept digging into it. Finally, a severe bump happened, killing one of the men outright. The proper longwall face was scarcely affected by this bump.

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[13] Reference of footnote 12.

Referring again to Mr. Rice's plan, if, on the other hand, it is found that the pillars being drawn are crushed by the accumulated pressure at this point, it would appear to me that the splitting of the adjacent outby pillars would be an extremely hazardous operation. These adjacent pillars must—if the pillars being drawn are crushed—be carrying a tremendous burden; splitting will reduce the area of pillar coal carrying this load and quite conceivably increase the stress per unit area of pillar to such a point that the pillars will burst.

Mr. Rice apparently is placing great reliance on the support afforded by rock-filled cribs, but unless the sandstone beds forming the roof are capable of considerable bending movement it would seem that these cribs will not afford the necessary rigid support. A rock-filled crib under pressure will yield at first fairly readily, and it is not until the crib has yielded to such an extent that the confined rock becomes compacted that real effective resistance is offered, and while these cribs will be of very material assistance in roof control—indeed, I doubt whether the pillars could be drawn successfully without them—it is doubtful whether they will be of assistance in overcoming the conditions causing bumps.

A better method of working would be to leave larger pillars without splitting them and draw the pillars from the goaf side, thus approximating more nearly to a longwall retreating method. It is also suggested that at certain intervals the spaces between the cribs should be filled with rock retrieved from the waste, thus forming support walls at right angles to the general line of retreating faces.

Mr. Rice, in discussing the bumps at Springhill, suggests that the narrow or ribbon-like width of opening, 500 ft., may be responsible for a cantilevering of the roof sandstone, thereby throwing an additional load on the coal adjacent to an extensive goaf. Some of the officials at this mine also hold this view but an argument against this is the case of the inbreak of water[14] coming from the flooded waste of another seam lying some 400 ft. overhead, caused, it is thought, by a longwall development, measuring only some 500 by 500 ft., breaking the overlying rocks; the thickness of the superincumbent sandstone at this point being 68 ft. If this extent of opening was sufficient to fracture the sandstone, how much more so would a 500-ft. width of opening contacting directly with an approximate width of 1000 ft. of goaf fracture a sandstone only some 26 ft. thick. It must be admitted that there is no positive proof that the water came through at this point, but as previous pillar and room work had been carried on under this flooded waste for nearly nine years without letting down the water, the assumption that the longwall work was responsible for the inbreak appears to be fairly sound.

G. S. Rice.—Mr. McCall asks whether the bump in a lower bed caused cracks to appear in an upper bed three or four weeks after the bump. There was no information that there was any bumping or jarring of the lower bed during this interval, but it is presumed that there was not or it would have been commented upon by the mine operators. I do not think that the rupture of rock by tension is necessarily accompanied by jarring or bumping.

Mr. McCall referred to a bump that struck a pillar not nearest the goaf but second from it. The pillar nearest the goaf was not crushed. Mr. McCall suggests that one of the conditions necessary to cause a pillar to bump must be capable of resisting a heavy pressure. I concur in this but when it is found, after a bump, that the main roof is not broken or materially bent down I believe it must be the result of a shock from compression failure in a rock stratum above, which strikes a blow of limited lateral extent, such as might result from failure of an arch, the buttress of which rests over the burst pillar.

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[14] T. L. McCall: Further Notes on Bumps in No. 2 Mine, Springhill, N.S. *Trans.* A.I.M.E. (1934) **108,** 41–60.

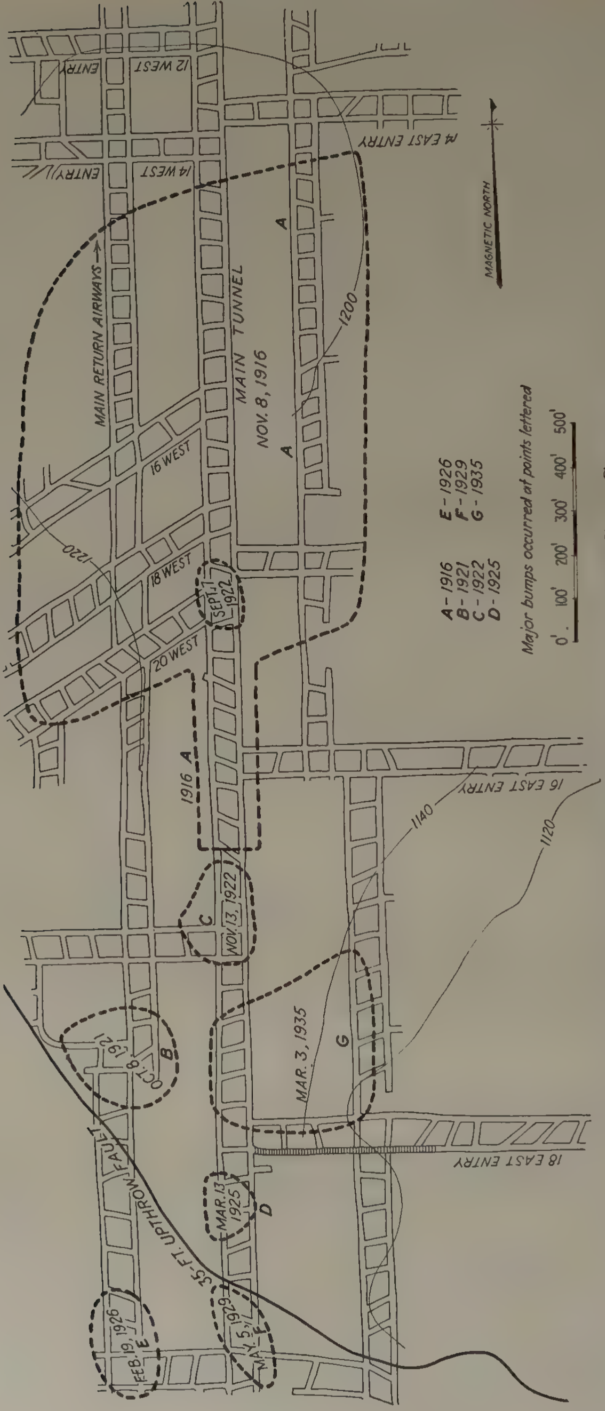

FIG. 8.—PORTION OF NO. 1 EAST MINE MAIN TUNNEL, COAL CREEK COLLIERY. Note size of pillars. Bumps occurred at points indicated by letters and dates.

B. CAUFIELD,* Michel, B.C. (written discussion).—The methods suggested by Mr. Rice for operation of the mines at Coal Creek, B.C., after his examination in 1916, were put into effect in the early part of 1917. Operations were extended eastwards and under heavier cover, when we encountered outbursts. Altogether we had over 300 of these ranging in size from a few tons to several hundreds, which gave off, at times, millions of cubic feet of methane.

On Aug. 28, 1928, a large outburst occurred, which killed six men and menaced the whole mine crew. There were 1300 tons of coal discharged and the volume of gas reversed a current of 30,000 cu. ft. of air per minute. Explosive gas extinguished a safety lamp as far back as 4000 ft. on the intake side of the outburst. It was estimated that ten million cubic feet of methane was discharged.

From 1916 up to the present there have been 42 major bumps, the recent one being on March 3, 1935, which did considerable damage to the roadways of the mine. There is no pillar work in this mine and extraction is about 16 per cent. Rooms are driven on the strike in series of three's, 15 ft. wide, with 50-ft. pillars between rooms and 200-ft. pillars between series; with the 200-ft. pillar crosscutted for ventilation every 300 ft.

An important feature in both outbursts and bumps is that there is usually some geological disturbance in close proximity to these occurrences, and I have come to the conclusion that they are caused by the release of ground stresses, which were formed by ground movement under heavy cover. On one occasion there was a serious bump in a pair of prospect entries at a point 1200 ft. from any other workings. After cleaning up and driving forward about 50 ft., we encountered a 14-ft. upthrow fault. On three other occasions there were bumps at places where the rock roof was brushed down leaving the sides or ribs in solid rock, and large pieces of rock were burst violently from the ribs, one piece weighing about 26 cwt.

It is worthy of note that a bump in Pendleton mine, England, occurred in a longwall mine close to the coal face, killing five men, and that this occurred in heavily faulted ground.

Previous to 1916 the No. 2 mine at Coal Creek tried an advancing longwall system of work at two different sections, to try to prevent bumps. One section had a suitable cap rock for pack walls and gob packing. The cost of operation, however, was too great and the system was abandoned. The other section was worked under a hard conglomerate roof and a system of cogging was tried, the cogs being filled with rock taken out of the floor in the roadways. The roof settled down and crushed the cogs into the floor, causing the floor to heave up around the cogs quicker than we could keep the roadways open for traffic. I am inclined to think that had we worked a system of retreating longwall, it would have been successful. For this reason I am in sincere agreement with Mr. Rice's suggestion of working some modified system of retreating longwall in the Cumberland field and using systematic supports of rock-filled cogs to support the subsidence in the gob.

Fig. 8 shows the position of the recent bumps at Coal Creek mine.

G. S. RICE.—The Crow's Nest Pass Coal Co. has had at its Coal Creek mines more serious experiences with violent coal bumps than any other coal mines of which I know, and in many instances they were accompanied by instantaneous outbursts of gas. It must be remembered that the coal underlies a high mining plateau, giving a cover rising over 2000 ft. above the coal and probably with unbalanced stresses. It is interesting that, in spite of some apparently adverse experiences, Mr. Caufield regards favorably some modified system of retreating longwall to meet the bump conditions.

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* Superintendent of Michel Mines, Crow's Nest Pass Coal Co.

Method of Eliminating Coal Bumps or Minimizing Their Effects

By J. F. Bryson,* Associate Menber A.I.M.E.

(New York Meeting, February, 1936)

Instantaneous outbursts of coal in underground workings have occurred frequently in various coal fields in Nova Scotia; British Columbia; Canada; South Staffordshire, England; and the states of Washington, Virginia, West Virginia and eastern Kentucky in the United States. Such outbursts in eastern Kentucky have been called "mountain bumps" because they are more likely to occur under mountain cover than elsewhere, but recently they have become generally known as "coal bumps."

The eastern Kentucky coal field has been troubled by these outbursts since 1923. Many men have been injured and killed, and the morale of the workmen in the vicinity of such outbursts has been affected to such an extent that mine production has suffered for a considerable period following each outburst.

Such phenomenal occurrence in any coal field naturally has led to discussion as to the cause and means of prevention. In the eastern Kentucky field, local mine officials, workmen, mining engineers, and many others have offered many ideas as to the cause and many methods of prevention have been suggested, and tried without success. Some of the plans were either practically or economically unsound and in general impossible to follow. To enumerate all the suggestions would require a large volume. One cause mentioned was that they might be due to gas under high pressure, as in Lower Silesian mines, and the remedy recommended was to drain the gas by drilling holes into a suspected area. A hole 30 ft. long was drilled into a 60-ft. pillar to allow the gas to drain off, but the pillar bumped the next day.

Realizing the importance of obtaining the best information available on coal bumps, the writer, with the permission of the Harlan County Coal Operators' Association, invited Mr. George S. Rice, Chief Mining Engineer of the U.S. Bureau of Mines, who had investigated bumps in several countries, to conduct an investigation of coal bumps in this field. Mr. Rice, by permission of the Director of the U.S. Bureau of Mines, accepted the invitation, and it was the writer's privilege, in company with

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* Director of Safety, Harlan County Coal Operators' Association, Harlan, Kentucky.

Mr. J. F. Davies, District Engineer of the U.S. Bureau of Mines, Norton, Va., to accompany Mr. Rice and assist him in his investigation. Mr. Rice's report and recommendations were printed by the U.S. Bureau of Mines[1] and sent to all mines.

The recommendations made by Mr. Rice were a straight retreat line with rock-filled cribs for a cushioned support of roof.

The Kentucky State Department of Mines and Minerals, under the leadership of the Chief Inspector, Mr. John F. Daniel, appointed an investigation committee, composed of two geologists and one mining engineer, to study the coal bumps and formulate some method for their prevention. This committee recommended that only first mining should be carried on, room pillars and chain pillars being left. By this means, the Committee believed, 73 per cent of the total property could be mined in safety without coal bumps, but at a sacrifice of 25 per cent of coal reserves.

Definition of Coal Bumps and Theory of Occurrence

Bumps in coal mines are sudden, unpredictable, violent bursts of coal from a pillar or pillars similar in effect to a great blast of explosives without the accompanying heat effect, usually accompanied or preceded by an air blast and great clouds of finely pulverized coal dust in suspension. Gas may also be released. Twice a gas ignition followed a bump in eastern Kentucky, caused in one instance by an open light and in the other probably by an electric spark.

Usually in a coal bump large quantities of coal are thrown off the burst pillar and the remaining part of the pillar, although still remaining on its foundation, is finely pulverized. The main roof is seldom broken or even cracked. The roof is sometimes stained brown, which the writer surmises indicates pressure on the pillar before the bump took place. Mr. Rice's suggestion is that it may be due to lateral sliding movement of the roof on the coal incidental to bumping. In one instance a sight line that had been painted on the roof showed a lateral movement of three inches.

In most bumps posts have not been broken. In some instances posts that had shown weight previous to the bump could be knocked out by hand, probably because the prop had been forced into the floor. This and the foregoing features indicate a violent vibration of the immediate roof where it is strong and has a certain amount of elasticity. Usually after a bump where the pillar has been crushed a space is found from one to three inches between roof and coal. Sometimes a large cavity is found in the center of the pillar, and in one outburst where half of the pillar had been moved laterally a distance of 6 ft. one man was found embedded between the top of the coal and roof while the other workman had been thrown a distance of 25 feet.

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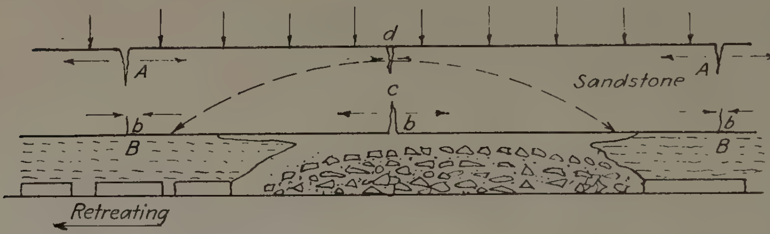
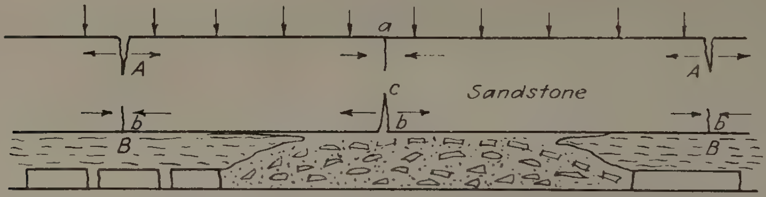
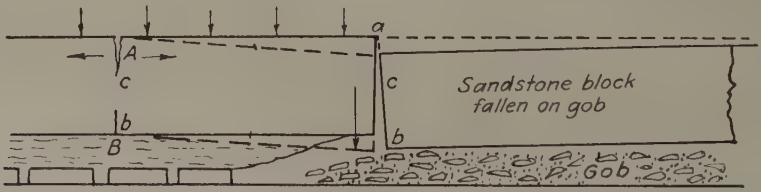
[1] G. S. Rice: Bumps in Coal Mines of the Cumberland Field, Kentucky and Virginia—Causes and Remedy. U.S. Bur. Mines *R.I.* 3267 (1934). Subsequently Mr. Rice prepared a general paper on bumps, which was presented at the meeting of the A.I.M.E. in February, 1935. See page 11, this volume.



a. Breaking of strong roof stratum unsupported in goaf, which, acting as a flat arch ruptures by crushing in the upper part and sends shock waves to supporting pillars.



b. Breaking of strong roof stratum unsupported in goaf, which acts as a beam but, as above, ruptures by crushing in upper part and sends shock waves, probably in curved lines, to supporting pillars.



c. Breaking of strong roof stratum when acting as cantilever partly supported in goaf by broken-off material or, in absence of this, is given a compressible support by filled cribs, rupture occurring by tension failure that does not produce shock waves because tensile strength of rocks is small.

FIG. 1.—THEORETICAL CONDITIONS AND NATURE OF RUPTURE STRESSES OF STRONG ROOF STRATA THAT FAVOR OR LESSEN LIKELIHOOD OF DESTRUCTIVE BUMPS IN CUMBERLAND FIELD. (*After G. S. Rice, Bur. of Mines R.I.* 3267.)

Mr. George S. Rice describes two destructive types of bumps, which he has termed "pressure bumps" and "shock bumps." The writer is in complete agreement with Mr. Rice in his definition of types essentially as follows: Pressure bumps appear to be due to the unit loading on pillars being too great for their bearing strength and where the coal and roof are strong the pillars are ruptured just as a small block of coal or rock is ruptured suddenly and with violence in a pressure machine. Shock bumps are due to the breaking of a thick, massive, rigid stratum somewhere above the coal bed, which causes a great hammerlike blow to be given to the immediate roof, which it transmits as a shock wave to the coal pillars (Fig. 1). The original blow may originate in one of several ways: (1) The sudden failure of an unsupported thick stratum acting as a flat arch when the span has become too great (200 or 300 ft. or more) and it yields by crushing in the upper part; the shock produced by the failure in compression of a strong stratum like sandstone is very great and the

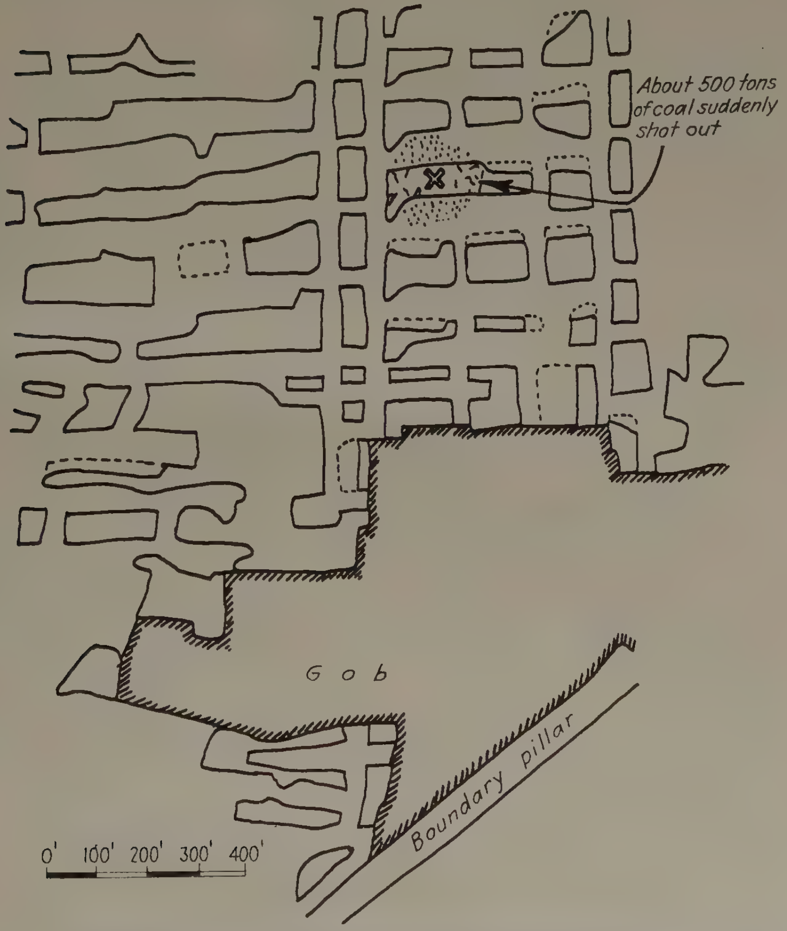


FIG. 2.—ILLUSTRATION OF FACT THAT THE PILLAR THAT BUMPS IS NOT ALWAYS THE ONE NEXT THE GOB. (*After G. S. Rice, R.I.* 3267.)

Cover approximately 1100 ft. of sandstone; 60 in. hard slate, 4 in. draw slate, 52 in. coal; hard bottom.

shock wave is then transmitted like an earthquake shock to the pillar supports; (2) a similar failure of an unsupported beam extending over the pillars and unfilled goaf; (3) the mass of rock falling upon a lower layer which previously has subsided. This distance through which the mass falls may be less than a foot, but the mass of rock plus the weight of the ground resting on it may weigh many thousands of tons, hence the foot-pounds energy imparted in striking will be very great[2].

It is not possible in any specific case to determine by which agency the shock wave reaching the pillars is caused. The path of the major component of the wave is conjectural, as there are too many unknown factors. The important thing is that a great blow is given on a pillar or a group of pillars, and it may not be the pillar nearest the gob or worked-out territory that receives the greatest shock. In one occurrence it was the fourth pillar from the gob that bumped (Fig. 2), and killed two men. From the observations made in the eastern Kentucky coal fields, "shock bumps" appear to be the principal if not the sole type of bump experienced. Where it has been the practice to leave small pillars, it has been observed that creep and squeezes have developed, which lead to pressure bumps.

Bump Area, Eastern Kentucky

Coal bumps have occurred with increasing frequency in the part of the Middlesboro syncline in Harlan County, Kentucky, south of the Poor Fork of Cumberland River, east of a north and south line through Harlan, north of Stone Mountain and in the western part of Wise County, Virginia. Nine coal companies, involving 14 mines in this section, have been affected. The mines are on Big Black Mountain and Little Black Mountain (Fig. 3).

Conditions Favoring Bumps.—Conditions favoring bumps are essentially as stated by G. S. Rice in *R.I.* 3267 (pp. 4 and 5), as follows:

1. Where there are strong, rigid rock strata immediately above the coal bed and hard strata predominate throughout the overburden.
2. A strong floor that will not heave readily under pressure.
3. A structurally strong coal, which in the pillar does not squeeze easily.
4. An overburden above 1000 ft. Bumps have occurred in 500 ft. of cover but generally speaking they are confined to 1000 ft. or more of overburden.
5. Faulty mining methods.

There is no doubt that faulty mining methods play an important part in creating bump conditions; that is: (1) pillars too small; (2) leaving projecting pillars behind in line of pillar withdrawal; (3) narrowing pillars to points; (4) drawing pillars on separate panels without any attention to a long continuous break line; but the method of mining adopted in the eastern Kentucky coal field is no worse than is found in the majority of coal fields. The unbalanced weight in mountain ridges is also a factor in creating bump conditions; that is, weight that is without lateral support.

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[2] A disastrous bump in Crow's Nest Pass coal field was attributed by G. S. Rice to such a cause. British Columbia Dept. of Mines *Bull.* 2 (1918).

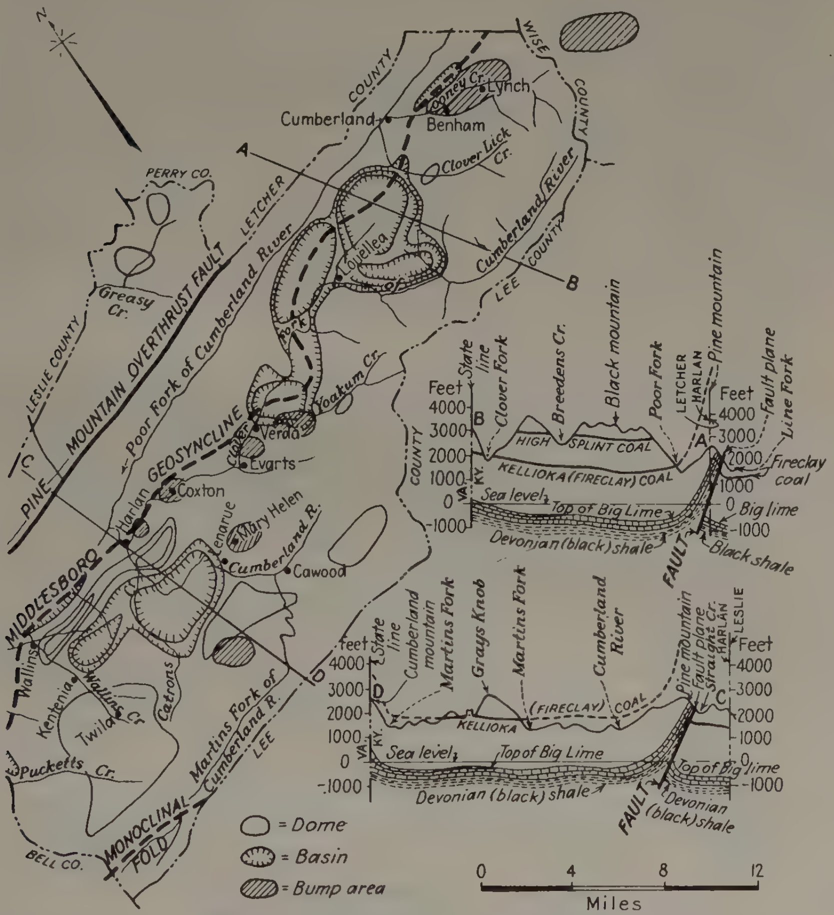

FIG. 3.—BUMP AREA.

Structure and Texture of Coal.—The coals of the bump area lie within the trough-shaped basin known as the Middlesboro syncline or basin; generally speaking, the coal beds are flat but dip gently from the foot of the upturned strata on either side of the syncline. Local dips and swags are present in all the mines. Sometimes a swag or depression is accompanied by a roll in roof. Rolls in roof have occurred without any corresponding depression in the bottom of the coal seam. Some of these rolls extend down into the coal as much as 3 ft., as in the section of the mine where remedial measures for prevention of bumps was carried out. The coal is extremely brittle and possesses a good hardness for bituminous coal. It breaks and shatters readily under a blow or strain. It is a dark, glossy black when undisturbed, but under pressure it shows the true dark brown color of bituminous coal. The face cleavages in the coal are well defined. Butt faces are also numerous and well defined.

The U.S. Geological Survey and the Kentucky State Geological Survey consider that the coals in the eastern Kentucky coal field belong to the Pottsville measures of the Mississippian series of the Carboniferous age. The Pottsville series in the bump area is made up principally of sandstone, shale and coal. Shales and their intermediate gradations occupy nearly the whole section, with sandstone predominating. Sandstone in the Pottsville series is of varying thickness, running from a few feet to over 100 ft. Sandstone overlying the coal is from 30 to 90 ft. thick, as shown by logs of prospect drill holes.

Remedial Measures

The mine where the remedial measures for elimination of coal bumps were tried operated the Harlan seam, worked by a room-and-pillar method. The cover ran from 1200 to 2000 ft. and the immediate roof over the coal was a massive, hard, sand rock estimated to be from 70 to 90 ft. thick. The average height of coal throughout the mine was 46 inches.

Various sections were troubled by sandstone rolls in the roof, which often extended down into the coal. Face entries were driven 1200 ft. apart, center to center. Butt entries were driven on 400-ft. centers. Rooms were 35 ft. wide on 70-ft. centers. They were kept in alignment by sights. Where room pillars were drawn the line of withdrawal was irregular. Pillars projected back into the gob.

Prior to 1931 coal bumps had occurred frequently on the pillar retreat line but no workmen had been caught. One man had been killed by a coal bump in 2 left off 3 face in 1931, just before the work in the entry was completed. On May 12, 1934, two men were killed by a coal bump on No. 9 pillar, 4 left off 3 face. This pillar was projecting into the gob.

The company called into consultation all the mine officials in the district, with the idea of gaining what information they could to eliminate coal bumps. Mr. Rice visited this mine in July, 1934, and recommended rock-filled cribs and straight withdrawal of pillar line. Another recommendation, made by a mine official, was that posts in groups be set around the pillar before withdrawal was started. The groups of posts consisted of four jack posts set together on 4-in. soles with 6 by 12 by 18-in. headers. It was believed that these would tend to minimize effects, if not prevent a coal bump.

The company agreed to try both plans. During subsequent visits to this area the writer observed that the groups of posts were of very little use in preventing or even lessening the effects of the coal bumps. On the other hand, it could be plainly seen that the rock-filled cribs did tend to lessen the severity of the bumps, although they did not prevent them entirely. The reason for lack of complete success was the irregular line of pillar withdrawal. It was indicated that if the pillar line were straightened and cribs placed on proper centers, it would allow the roof to subside gently without rupture of the immediate strata, which gave promise of the elimination of the coal-bump hazard.

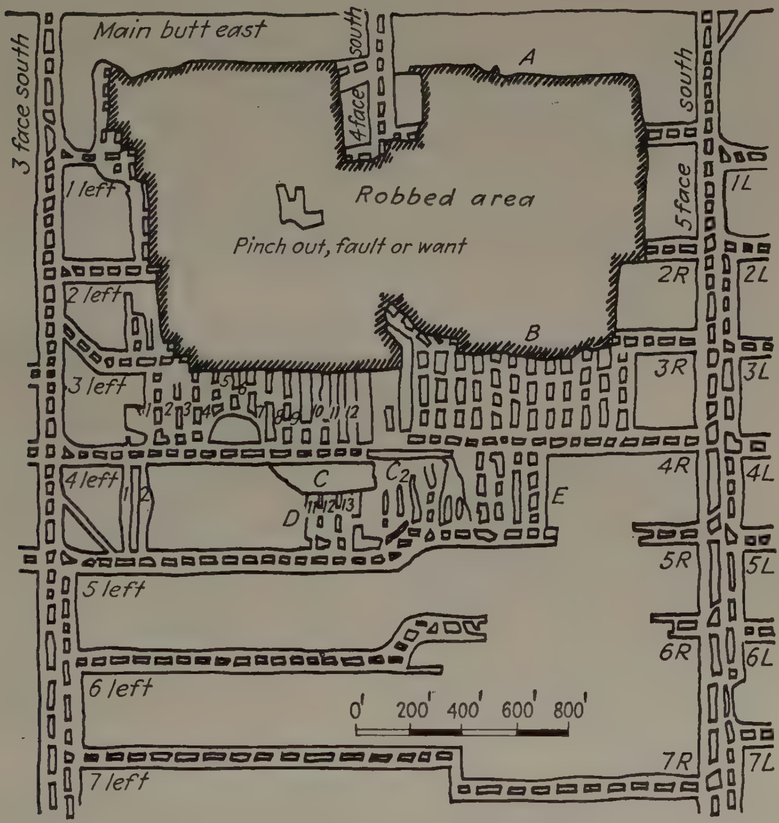

FIG. 4.—SECTION OF MINE MAP, BUMP AREA.

Small bumps occurring regularly did a great deal to impair the morale of the workmen, so the company was urged to straighten the pillar line and start a broad or long retreating face with rock-filled cribs as supports behind it. Many times this recommendation was made but the officials were afraid to try it and argued that it would not work in the Harlan seam. However, after very thorough deliberations, the decision to try out a long face was finally made.

Place Selected for New Plan

After gaining the sanction of the company officials to try a long face with cribs, it was but natural to select a place where bumps had occurred with regularity. The section between 4 and 5 left off 3 face (Fig. 4) answered the purpose. Five men had been killed in this area from coal bumps. One, two and three entries off 3 face had been pulled back to the barrier pillar. A "pinch-out" fault or "want" approximately 200 ft. square was situated in the robbed area between one and two left. A robbed area of 1600 ft. west to east (*A* to *B*) bordered the place of experiment at the ends off the rooms of the 4 left, a 400- by 50-ft. slab had been worked as a longwall face inby No. 8 room on 4 left side *C*. Inby No. 12

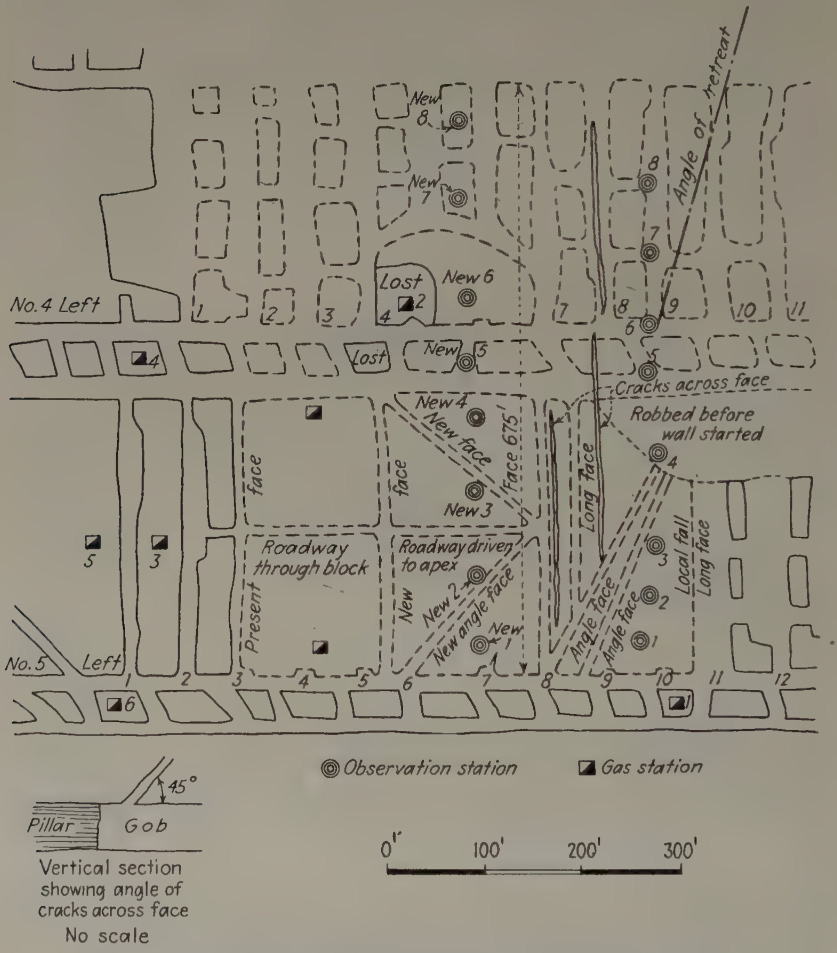

FIG. 5.—SECTION OF MINE MAP SHOWING DETAILS IN BUMP AREA.

room 4 left another place 300 ft. wide had been worked as a long face *C*2. Fourth left had been driven through a series of sand-rock rolls. From the 5 left, rooms 11 to 22 (*D* to *E*) had been driven up to 4 left. These rooms were bumping very badly and had driven all the workmen out; not a man could be found who would willingly work any of these rooms. It was agreed to start the long face in No. 11 room (at *D*) off 5 left, and at the same time drive across from 4 left; the fourth left side to work on an angle until fifth left face retreated 65 feet.

While this work was going on the pillars on 4 left side were to be worked on a straight line corresponding to the line of withdrawal of the long face. The wall to start with would be 175 ft. on the 5 left side and gaining every day on the 4 left side until the total long face was 350 ft. Then if the pillars on the 4th left side were worked on a straight line, the long face would ultimately be 675 ft. long.

It was felt by all concerned that if coal bumps could be eliminated in a section such as this the solution of the problem would be on its way. The roof cover at this particular section was 1400 ft. with sandstone predominating. The experiment was started on Jan. 15, 1935. (See Fig. 5.)

Cribs

It was decided to use ordinary mine post timbers of chestnut for cribs with a few beach and larch timbers. These were 3 ft. 6 in. long and not less than 6 in. thick, of the split-timber type. Twenty-five posts were allowed to a crib with an 8-point bearing surface. As rock material for filling the cribs was very difficult to obtain at the mine, it was decided to try fine coal for the packing of the cribs. When completed the cribs were 42 in. square, and to start with, cribs were set on 20-ft. centers (Fig. 6).

Convergence and Subsidence of Roof

Eight stations for recording information on convergence of roof and floor were established, but before the stations could be set up it was necessary that sufficient coal should be extracted to enable this to be done. The height of the coal where the stations were to be placed varied from 41 in. at No. 8 station to 52 ft. at No. 1 station. All that was hoped for from such convergence records was to obtain the rates of subsidence of roof and rise of floor in relation to the position of the coal face at time intervals. Halbaum and Winstanley, and other British investigators, in their respective papers on roof control, maintain that complete extraction of any coal seam induces "weighting," hence more or less convergence of roof and floor, owing to compression of the coal immediately in advance of the area of extraction. It is interesting to note that during the whole time of this experiment only once was convergence of roof and floor detected in the coal face, and at no time in advance. This appears to indicate the great strength of the roof.

The roof in this section being of hard, massive sandstone, considerable area had to be extracted before any signs of subsidence started at or near the face. Once it did start there was almost a continuous movement. Posts that were left between the cribs broke readily before the cribs took up the weight. Once compression of the cribs started, the movement was steady.

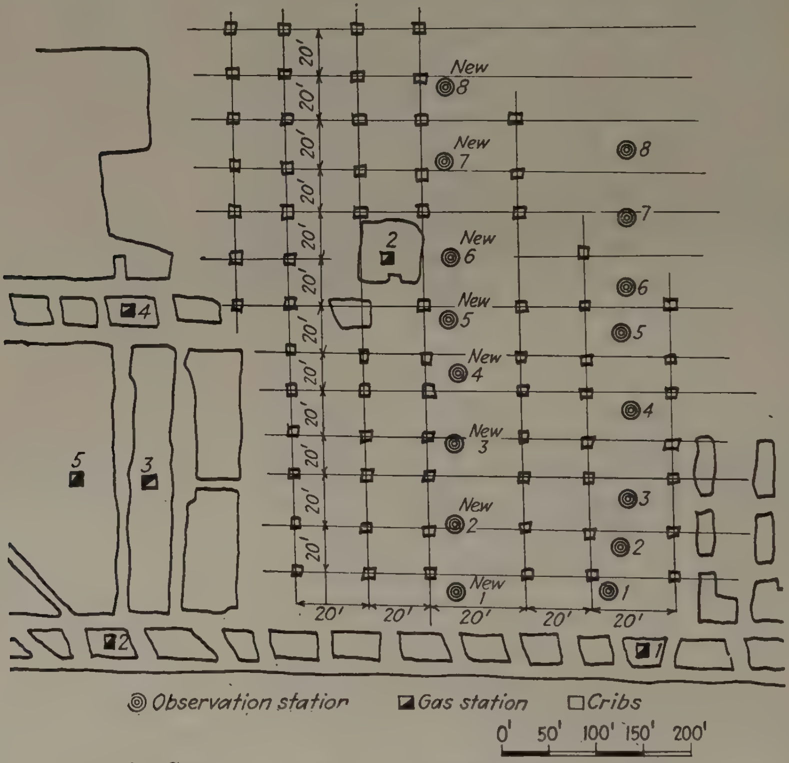

FIG. 6.—SECTION OF MINE MAP SHOWING METHOD OF CRIBBING.

After the first hairline cracks in the roof appeared they gradually widened, one side falling back on an angle of 45° towards the gob. Widening out from hairlines, these cracks became 16 in. wide without a fall. The subsidence with compression of cribs is indicated in the diagram in Fig. 7, these measurements being taken every five or six days. The device used for measuring subsidence was the two-stick slide calibrated in inches, commonly used on longwall workings.

The stations were placed on Feb. 1 and the first measurement was taken on Feb. 7. It showed that a decided movement of roof subsidence had started. The measurements at stations (Fig. 6) were taken up to

May 10. After that date it was impossible to reach the stations, owing to the great subsidence of roof. New stations were then selected and the convergence measurements carried on. The roof settled gradually all the time until it reached at the successive points the subsidence shown in the convergence chart (Fig. 7). After that it was stationary for a considerable time.

The floor in the mines is strong and as there was no perceptible rise of floor, such as is said to take place in certain longwall mines in England, the lessened height in the excavation was considered practically all due to roof subsidence.

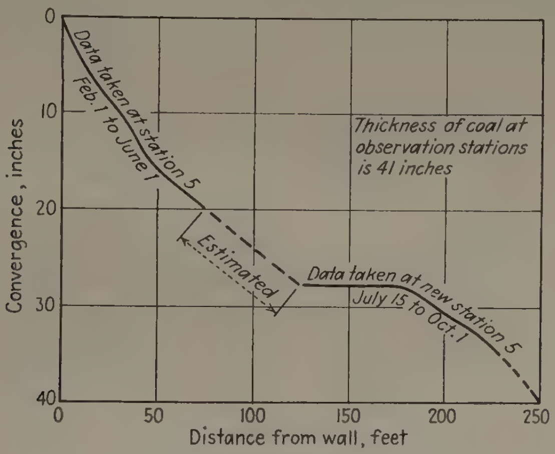

FIG. 7.—CONVERGENCE OF ROOF.

The convergence varied from ¼ in. at station 1, to 1½ in. at station 8, which was 600 ft. from station 1. Stations 1, 2, 3 and 4 were on the 350 ft. of long face; stations 5, 6, 7 and 8 extended from 4th left entry back into the area where 4th entry pillars had been extracted. The absence of pillar support accounts for the rapidity of subsidence in this vicinity. When the compression of the cribs had reached a certain point, a pool of water formed around the bottom of each crib, which evidently was squeezed from the timber, although the posts that had been built into the cribs were to all appearances dry when they were brought into the mine.

UTILIZATION OF PRESSURE ON COAL FACE

On starting the long face of 175 ft., it was necessary to undercut the coal by machine. This method was continued until the subsidence of roof was carried forward to utilize the pressure exerted by the subsidence to assist in the working of the coal. Then the coal-cutting machines were withdrawn and the coal obtained by pick work without the aid of explosives. The dangerous part here was to allow just the right amount of subsidence to prevent the weight from being flung forward on the long face, which would interfere with its working. Once the pressure of the subsiding roof was in active operation on the coal, its proper application as a rending and fracturing force was one of the most important aids in working the long face.

As the face advanced the superincumbent strata gradually settled on the cribs. As it settled it was usually accompanied by cracking, grinding, and thuds of roof, but no cracks or breaks were noted. The workmen not accustomed to long-face work usually left in a hurry when the roof started to grind. Much effort was expended in teaching the workmen that there was no immediate danger when the roof was settling down on the cribs.

Issuance of Methane Following Bumps

Many times the writer has been at the scene of a coal bump almost immediately after the bump has occurred. The methane indicator was used very frequently at such times and the highest range of methane found in the air after a bump was 2.7 per cent; usually it was far less, ranging from 0.6 to 1 per cent. Later it was found that the high percentage was in air that had been short-circuited, therefore the high percentage did not prove of any indicative value in this case. However, the evidence gathered from a large number of bumps does show that methane is liberated in considerable quantities during a coal bump. What is the actual amount of methane released by a bump would be impossible to say, as it would depend greatly on the amount of coal under pressure.

To gain some knowledge of the amount of gas given off under pressure, 6-ft. holes 1½ in. in diameter were drilled in some chain pillars adjacent to the long-face experiment. A pipe was placed in each hole and tightly tamped with clay. A screw cap was placed on the end of the pipe and tests were taken at intervals. The holes drilled in the large block of coal forming the long face did not show methane in any particular quantities, but the holes placed in the chain pillars did show it. At one time an explosive mixture was found. While the record of methane found in the pillars was high (Fig. 8), it did not materially increase the amount of methane in the return airway. This can be accounted for by the splendid ventilating current circulating this section. However, the fact that sufficient gas was found in the pillars under pressure to form an explosive mixture does denote that methane in considerable quantities is liberated by a coal bump, but probably not sufficient to be a factor in bursting the coal as found in a few coal fields of the world with special geologic conditions[3].

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[5] See papers by G. S. Rice, P. A. C. Wilson, and others on Instantaneous Outbursts. *Trans.* A.I.M.E. (1931) **94**, 75–136.

Development of Remedial Work

After the long face was started the writer, in an advisory capacity, visited the long face twice a week during a period extending from Jan. 15 to March 7. While the 5th left side was producing from 75 to 100 three-ton cars daily, the local officials did not wish to start pillar extraction on the 4th left side, because of their fear of coal bumps, and the place that the writer thought should be driven from 4th left side to meet 5th left, and open face out to 350 ft., was not started. Such was the position on the morning of March 7, when the officials called and said they had "forward flung" weight on the coal face. This condition, the writer thought, was due entirely to irregular methods of extraction and absence of essential cribs.

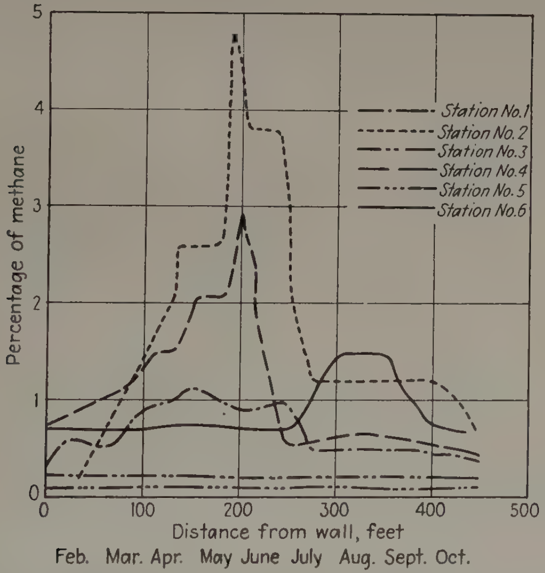

Fig. 8.—Percentage of methane observed at stations.

Cribs were then built on their proper 20-ft. centers and a line of breaker props on 3-ft. centers 70 ft. long was set parallel with the coal face and 75 ft. from it. Two cribs in the goaf, which were 100 ft. apart, were shot out, allowing the weight to fall in the gob up to the breaker line and taking it off the face of the coal.

Seeing the conditions and being afraid that bumps might occur on the No. 12 chain pillar and No. 8 room pillar on the 4th left side, the writer reported the matter to the officials of the company. They agreed to follow his recommendations and he arranged, by permission of the Coal Operators' Association, to spend at least 6 hr. every day on the long face until July, 1935, a period of nearly five months.

As he feared, No. 12 pillar, 4th left side, bumped on March 16 and No. 8 room pillar bumped on March 22. Fortunately, no one was injured, although the shock of the bumps was felt at the drift mouth. Both bumps were very large; however, a valuable lesson was learned from these two bumps, which was that the long face, with cribs systematically set up behind it as it advances, should be expected to support the weight of the roof.

On March 27, No. 12 chain pillar was completely extracted and No. 8 room pillar on March 29. On March 30, 4th left side had also cut through and for the first time since it started the long face was 350 ft. Arrangements were then made to start No. 11 chain pillar off 4th left and No. 7 block off 4th left. The fifth and sixth rooms from 4th left side were not driven from that entry (Figs. 4, 5 and 6), because of sand-rock rolls in the roof and bottom. The aircourse of 4 left and No. 11 chain pillar was also troubled with these sand-rock rolls. Such a condition as this was conducive to coal bumps and the greatest care had to be taken in the working of these particular pillars.

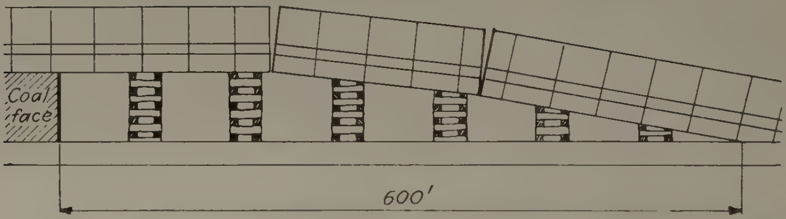

FIG. 9.—SECTION THROUGH FACE SHOWING REGULAR LINE OF BREAKS AND CRIBS.

On April 2, thin, hairlike cracks in the roof were noticed on the long face. These cracks ran parallel to the line of the face. On April 3, they had developed into a fracture of the roof end and there was a general subsidence on the cribs of from 3 to 5 in., the break gradually widening and falling back toward the gob. There was no fall and seemingly no danger that a fall would occur. The subsidence of roof, however, small as it was, caused the abandonment of the face. This was entirely due to the inability to get cars along the face. The roof subsided on the cribs from 3 to 5 in. directly in front of observation stations 3, 4 and 5. As will be noted in the convergence chart, the height of coal at these points was 41 in. (Fig. 7), which meant that instead of the foregoing measurements, there was 37 or 38 in. and the height of the mine cars on the rail was 37 in. (Fig. 9).

New Face Driven

On April 5, it was decided to drive across an 18-ft. wide place, starting to drive simultaneously from 4th and 5th left; to work three shifts continuously until through; two lines of cribs to be used. This wide place was started on April 6 and was cut through on April 13, when 600 cars were loaded out. When the first break occurred, it was thought that the number of cribs was insufficient, therefore the crib centers were reduced to 15 ft.; but this shortening of crib centers was quickly found to be wrong, and old centers were resumed. After work for two days on the new face, the hairline cracks in the roof again appeared, showing up in line with the face.

On April 15, to prevent a repetition of the former trouble and having to abandon the face, the ends of the wall on both 4 and 5 left entries were driven forward on a 45° angle, making the face the shape of a rough inverted V (Fig. 5). This idea was adopted because the cleavages in the rock strata were conformable with the face cleavages in the coal and it was thought that in leaving solid coal supporting at least part of the rock cleavage these breaks would not disturb the wall. Another point was that an area 250 by 1200 ft. had been extracted and the first longwall break was certainly due. Care had to be taken that this fracture should not come and cave in the whole face. The new line of face was successful and work was carried on for a considerable time. The running time of the mine was now considerably curtailed and under pressure of other duties the writer handed over this experiment to the local mine officials as of July, 1935, thinking there would be no further trouble, as the face was then progressing nicely.

On a return visit in August he found that the apex of the angle was trailing back into the gob nearly 100 ft. The part of the long face next to 4 and 5 left entries had been advanced that distance without touching the apex of the face. The pillar line on 4th left was irregular because a drawslate running from nothing to 2 ft. thick had shown up on 4th left side, also sandstone rolls and faulty coal, and the line of least resistance had been followed. While the officials were still afraid of the long face, they freely admitted that it stopped coal bumps and when nothing was wrong the wall would produce large quantities of coal, but that when something went wrong too many men were cut off from production.

It was then suggested, in order to remedy both troubles, that a narrow place be driven across the wall, starting from both sides. In the center of the wall, or 175 ft. from the entries, a narrow room would be driven up through to the apex of the V and this coal taken out. This would eliminate the heavy drawslate and would also take the coal out of the gob. Also, to go back to No. 1 room and drive a room up through the center of the block, so that if production were cut off by any trouble at the face there were three sides on which the block could be attacked. These narrow rooms were started on Sept. 18 and cut through on Oct. 1, and the extraction of the coal at the apex was started. The coal at the apex of the V was taken out both ways, the right side of the apex inby the face being driven parallel to 5 left entry and the left side being driven across parallel to the 4 left entry. All the coal was recovered and the straight face was resumed.

The narrow entry driven through the center of the block proved itself of great value. It was not only a safety measure, it could be used for haulage also, and if anything did happen to tie up the face and stop men from working they could be placed on one side or the other of the block to work until the face obstruction was removed. In this way production was always held at a peak. By cutting the wall into blocks, there was not only a means of safe exit for workmen in case of danger, but also one of the most efficient methods was formed for coal production. It has worked so well that the officials have adopted it in the other two walls that are following the experimental wall. These two walls were formed after the experiment had started. They were driven across 100 ft. wide, cut through to adjoining entries and started back on a long face. Cribs of timbers with 20-ft. centers packed with small coal are used.

Manner in which Roof Breaks in Long Face

The way in which the roof breaks in a long face with systematic cribbing is best indicated by a vertical cross section, Fig. 9. This section shows actual conditions in the long face of the 5th left, where the chain pillars are left and take some of the pressure.

The strong sandstone roof cushioned by the cribs evidently has broken by tension, a highly desirable way because, as stated by Mr. Rice (p. 23): "The tensile strength of rock is relatively low as compared with its great compressive strength; therefore in a rock stratum, acting as a cantilever projecting out over the pillars, the break would be by tension in its upper part . . . , an action which it is conjectured would not produce violent shocks."

Conclusions

The fact that the experiment, despite local handicaps, was carried on successfully for 11 out of 13 months speaks well for the method of long face with systematic cribbing. With all the difficulties experienced the wall has retreated 550 ft. without a sign of a coal bump; also, the pillars have not bumped since a straight retreat line was obtained. The success of the long face has restored the morale of the workmen. They are now confident that a long face with cribbing support will not bump.

To sum up, an area has been mined without occurrence of bumps where formerly they were almost a daily occurrence. The distance retreated was 550 ft.; the broadness or width of the face at one time was as much as 675 ft.; the running time from Jan. 15, 1935, to Feb. 15, 1936 (13 months) without a coal bump or injury to a workman engaged during this period, should prove the value of the method as a means of eliminating or minimizing coal bumps.

ACKNOWLEDGMENTS

The writer wishes to gratefully acknowledge the assistance received from Mr. D. J. Jones, State Geologist; Mr. N. M. Wilder, Field Geologist, Kentucky Department of Mines and Minerals; from Mr. Joseph F. Davies, District Engineer, U.S. Bureau of Mines, Norton, Va., in collecting data; and from Mr. George S. Rice, Chief Mining Engineer, U.S. Bureau of Mines, on many occasions and especially in reviewing this paper.

Subsidence at Merrittstown Air Shaft Near Brownsville, Pennsylvania

By F. W. Newhall,* Uniontown, Pa. and L. N. Plein,† Pittsburgh, Pa.

(New York Meeting, February, 1933‡)

During the latter part of the year 1931, the Republic mine of the Republic Steel Corporation, at Republic, about 4 miles south of Brownsville, Pa., was mining coal along four rib or fracture lines. One of these lines, the west butt rib line, was retreating towards a concrete-lined air shaft, 320 ft. deep, which had been sunk in 1911 for ventilation purposes. With the normal rate of mining at that time, it was expected that this rib line would pass the shaft in three or four months and the shaft would then be abandoned and allowed to cave. Realizing the opportunity presented for the study of an unusual case of ground movement, the question of conducting the study was presented by the junior author to the chairman of the Institute's committee on Ground Movement and Subsidence, Mr. George S. Rice, who presented the matter to Mr. G. H. Morse, General Manager of the Northern Mines, Republic Steel Corporation, and he agreed to cooperate with the Bureau of Mines in making the study, the results of which are presented in this paper. Acknowledgments are due to Mr. Morse for his full cooperation in this work and to Mr. G. S. Rice and Mr. J. W. Paul, both of the U. S. Bureau of Mines, for their advice and guidance during the course of the work. The authors are indebted to Mr. B. F. Herbert, mining engineer, Northern Coal Mines, Republic Steel Corporation, who assisted in all the surveys, and Mr. W. H. Smith, draftsman, U. S. Bureau of Mines, who prepared all illustrations for publication.

Geologic Conditions

The shaft is at Merrittstown, which is about one mile northwest of Republic. The topography in the immediate vicinity of Merrittstown consists of rolling country dissected by the meanders of Dunlap Creek and its tributaries. The surface slopes of the hills are gentle and their average gradient is less than 6°. The area at the point where the study was made lies on a hillside sloping towards the southwest, the gradient varying from 1° to 14°. Fig. 1 shows contour lines at 5-ft. intervals on

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‡ Presented again for discussion at the New York Meeting, February, 1935.

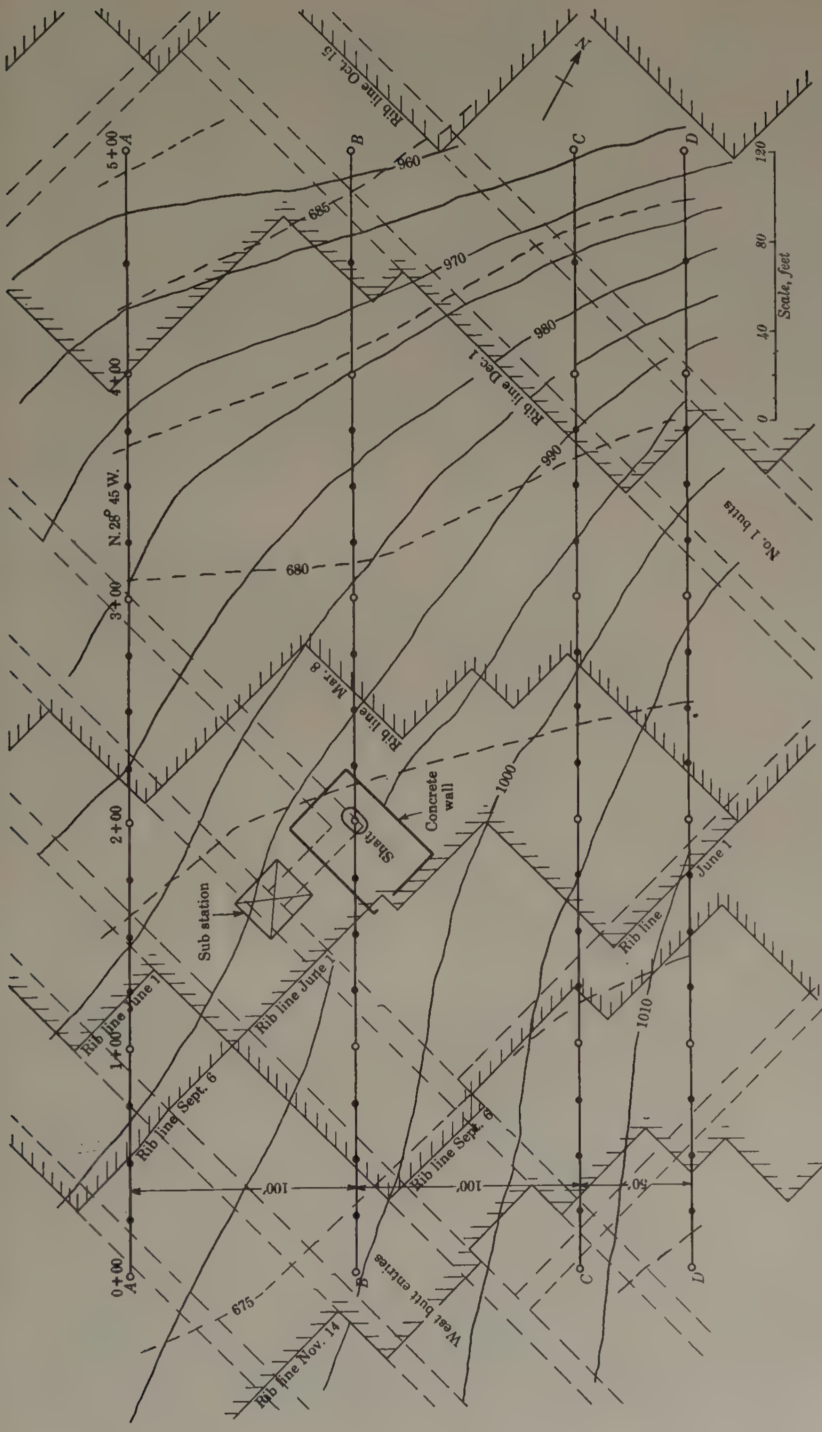

FIG. 1.—LOCATION OF MONUMENTS, SHAFT, SHAFT BUILDING AND RIB LINES FROM OCT. 15, 1931 TO NOV. 14, 1932.

Dec. 1, 1931. The Monongahela formation in this area is 370 to 400 ft. thick, and since the maximum cover in the area studied is 342 ft., we are concerned only with the strata of this formation. Structurally, the rock strata in this part of the Appalachian area are practically level but have a slight dip to the northwest; however, there are local structural folds in the region which reverse this general dip. Contour lines on 2½-ft. intervals, representing the bottom of the Pittsburgh coal bed, indicate that the coal lies practically level (Fig. 1).

From Fig. 1 the depth of cover at any point may be determined by comparing the elevations of the contour lines on the surface and on the coal bed. The cover at the shaft is 320 ft., the maximum cover is 342 ft. at station D-0 + 00 and the minimum cover is 269 ft. at station A-5 + 00. The average cover for the area bounded by the *A* and *D* lines and the zero and 4 + 00 stations is 318 feet. The word "cover" in this connection means the vertical distance from the point on the surface where subsidence occurs to the bottom of the coal bed at the point of extraction.

The Pittsburgh coal bed at Merrittstown varies in thickness from 90 to 98 in. but only 80 to 91 in. is mined, some top coal being left to support the roof. The average thickness mined is 85 inches.

No log showing the character of the rocks penetrated in sinking the Merrittstown air shaft was available, but two records of holes drilled about a mile from it were obtained and are given in Fig. 2. Drill hole A is 5735 ft. S. 60°30′ E. and drill hole B is 6765 ft. N. 70°32′ E. from the air shaft. These two logs were compared with records furnished by various coal companies within 3 to 5 miles of Merrittstown and the sections show sufficient similarity to indicate that holes A and B of Fig. 2 are representative of the strata at Merrittstown.

The rocks can best be described by quoting from the Masontown-Uniontown Folio of the U. S. Geological Survey, page 9:

> The rocks of the Monongahela formation are varied, but on the whole they are prevailingly calcareous, . . . The formation contains locally heavy beds of sandstone . . . The Pittsburgh coal, at the base of the formation, . . . is usually overlain by shale, sometimes fine and argillaceous, but more commonly stiff and sandy . . . Lithologically the most important member of the formation is the Great Limestone, which in places attains a thickness of 140 feet and occurs about 120 feet above the Pittsburgh coal. This bed is variable in composition and is never solid limestone. Frequently it may be divided into an upper and a lower division separated by shales and sandstones. The lower division probably has an average thickness of from 60 to 80 feet, and it is generally composed of alternating bands of limestone and calcareous shale. In places there are beds of solid limestone 10 to 12 feet in thickness, but such occurrences are rare. The limestone beds are usually less than 2 feet thick, but they are generally irregularly bedded and not good for quarrying purposes. The Great Limestone is particularly well developed in the northwest corner of the Masontown quadrangle, where it is exposed in all of the ravines leading down to the river. (Merrittstown is in the north central section of the Masontown quadrangle.)

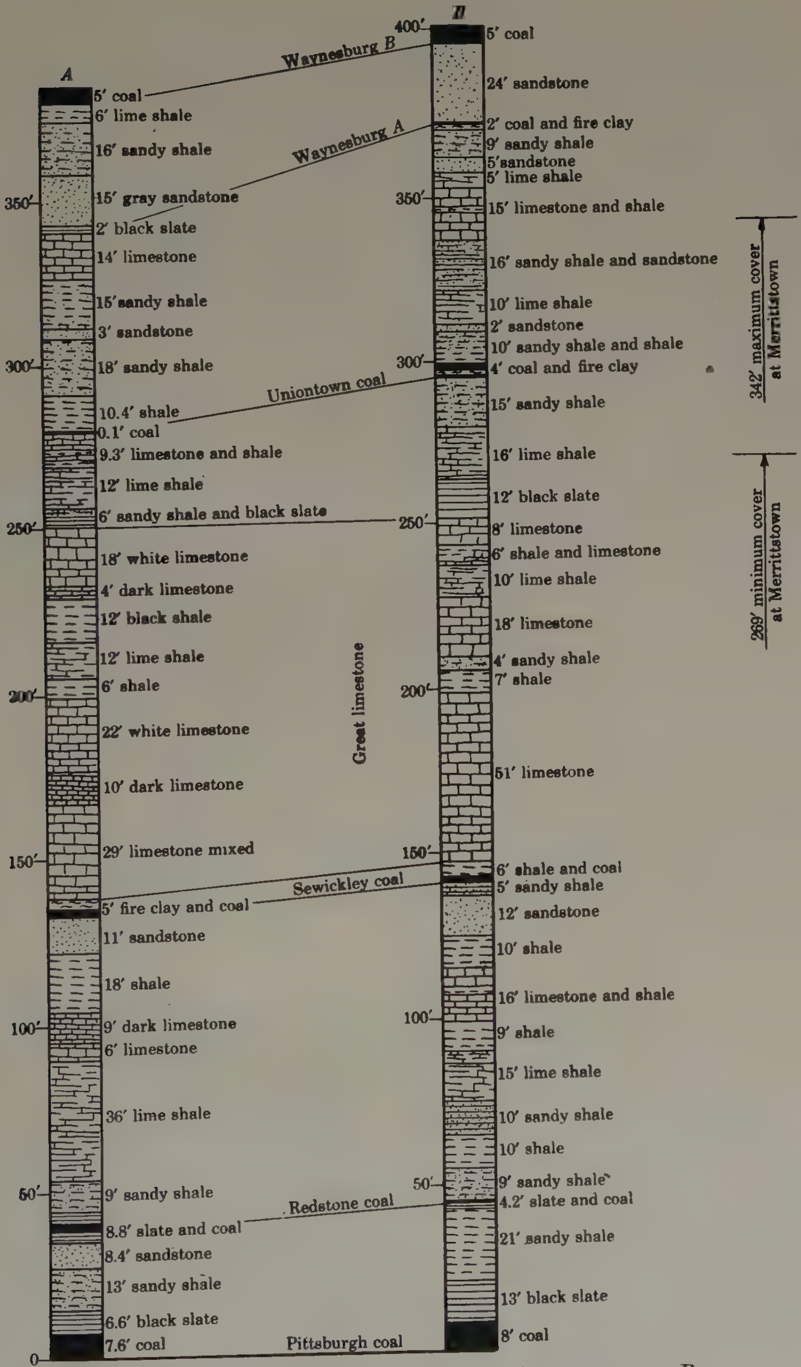

FIG. 2.—DRILL-HOLE RECORDS NEAR MERRITTSTOWN, PA.

METHODS OF MINING

The room and pillar panel system of development is followed, and the rooms are driven at right angles to the main coal cleats. All butt (or room entry) headings are driven 12 ft. wide on 50-ft. centers. Rooms are 13 to 14 ft. wide spaced on 100-ft. centers, leaving pillars about 87 ft. thick. Room crosscuts are driven also on 100-ft. centers, making pillar blocks 87 ft. square; thus, 25 per cent of the coal bed is extracted by advance mining. What is known as the Connellsville system of pillar extraction is used in the recovery of the room pillars. The general angle of retreat is 45° with respect to the face and butt entries; in detail, however, the extraction line commonly spoken of in the district as a "rib line" is in rectangular steps. The length of rib lines varies but the west butt rib line throughout the period of observation was 1500 ft. long.

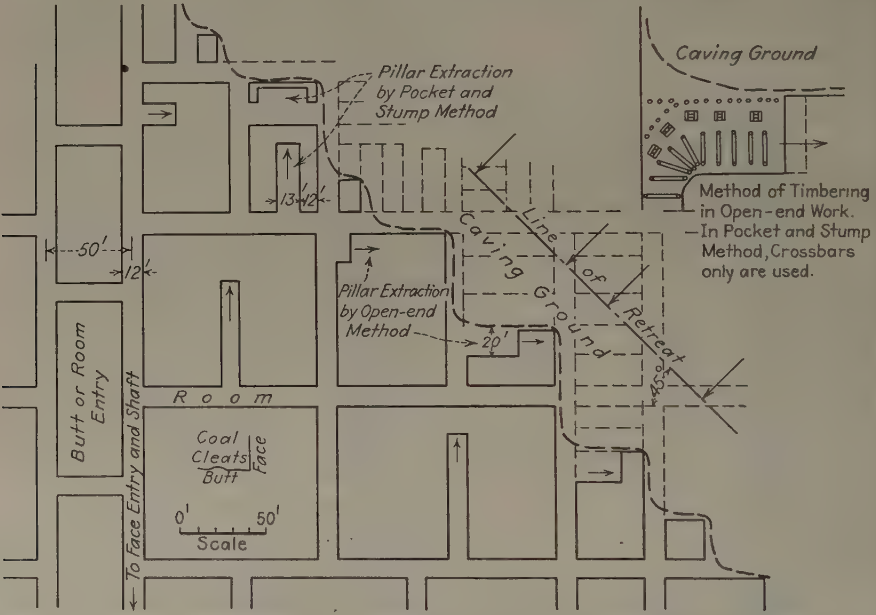
Pillar Extraction by Pocket and Stump Method
Caving Ground
Method of Timbering in Open-end Work. —In Pocket and Stump Method, Crossbars only are used.
50'
12'
13'
12'
Butt or Room Entry
To Face Entry and Shaft
Room
Coal Cleats
Face
Butt
0'
50'
Scale
Pillar Extraction by Open-end Method
20'
Caving Ground
Line of Retreat
45°

FIG. 3.—PILLAR EXTRACTION BY POCKET AND STUMP METHOD AND BY OPEN-END METHOD.

Before February, 1932, pillars along the retreat line were extracted by the pocket and stump method, which consisted of driving a pocket 13 ft. wide with a coal "fender" or "stump" 12 ft. wide against the gob. After the pocket is mined, the fender is extracted, timbers recovered and a roof fall made. Beginning in February, 1932, the open-end system of pillar extraction was followed. In mining by the open-end method, the 87-ft. square pillar block is split by driving a crosscut, thus forming two blocks 36 ft. wide and 87 ft. long. These are then mined open-end across the short side with each block worked in step with the general pillaring line. In this case no coal fender is left against the gob, but close timbering, consisting of posts, crossbars and cribs, is used to support the roof until the open-end face, which is 19 to 21 ft. wide, has been carried completely across the coal pillar. The timber is then removed and a roof fall made. Either method has the same effect with respect to subsidence, because in both methods the roof is supported, in one method by timber only and in the other by coal and timber, until the strip of coal, 19 to 25 ft. wide depending upon the method, is extracted, and then a fall is made so that the roof can break back to the solid block of coal. Fig. 3 illustrates the details of pillar extraction.

Field Methods in Determining Subsidence

Monuments and bench marks were established in November, 1931. Since the theoretical angle of retreat is 45° with respect to the west butt entries, the survey lines were placed at right angles to this theoretical line of retreat. Fig. 1 shows the position of these lines with respect to the shaft. The *B* line passes through the shaft with station 2 + 00 at the shaft center. On each line a monument was set every 25 ft. up to the 4 + 00 stations and then on 50-ft. centers to the 5 + 00 stations. These monuments consisted of 2-in. iron pipe, 24 in. long, sunk a few inches beneath the loose surface soil. A wooden plug was driven in the pipe with a nail for a center point. Bench marks and back sights were established well beyond the influence of any ground movement. Levels were run every two weeks, and, beginning Jan. 11, 1932, the horizontal movements, both longitudinal and lateral, were measured with transit and tape on all lines every four weeks.

Two plumb lines were hung in the shaft, one at station B-1 + 96.00 and the other at B-2 + 04.00. The shaft walls were marked every 10 ft. in depth with a spot of paint opposite each plumb line. The distance from the plumb lines to these paint marks was measured every two weeks from a free swinging cage lowered and raised by an electric hoist. At the shaft bottom two reference marks for each plumb line were painted on the concrete shaft walls. The distance from each plumb line to its reference marks was measured at the time of each survey to calculate the movement of the plumb line and to correct the measurements made from the plumb lines to the shaft walls. Throughout the period up to March 1, 1932, during which shaft measurements were made, there is no question but that these four reference points remained stationary; that is, they had not been affected by the approaching ground movement.

With reference to accuracy of surveys, it can be stated that all measurements by tape were accurate to 1 in 7000 or better. Measurements made in the shaft may be in error ±0.04 ft., because of the manner in which it was necessary to work. All leveling surveys started and finished at the same bench mark and errors of closure are available. All sights were read to 0.001 ft. on turning points and to 0.01 ft. on side points. The closing errors varied from 0.068 ft. to nothing. The average error per survey was 0.016. All elevations were balanced for each survey before being used. Only the surveys of Dec. 1, 1931, and Feb. 8 to Nov. 14, 1932, were used in the calculations. The average error of closure of these 22 surveys was 0.011 ft.; of these the two highest errors of closure were 0.032 and 0.030 ft. and all the others were 0.021 ft. or less. The closing error of the first survey made to determine the initial elevations was 0.020 ft. This degree of accuracy was considered to be satisfactory, because successive surveys every two weeks gave a check on leveling errors of previous and subsequent surveys, and because of the many factors of much greater variability involved in a study of subsidence.

Angles of Draw

The angle of draw is defined as the deflection from the vertical of a line drawn from the point of extraction to a point on the surface showing measurable subsidence. To determine the angle of draw, some value of initial subsidence must be assumed which definitely indicates that subsidence has started and that any apparent sinking, as indicated by the surveys, is not due to leveling errors. At 49 different stations the first measurable indication of horizontal movement averaged 0.03 ft. and the accompanying subsidence averaged 0.05 ft.; and these values are based only on surveys in which the closing error was 0.021 ft. or less. Therefore, the value of 0.05 ft. was used in determining the initial angles of draw, because, based on the knowledge of conditions, it is certain that a drop of 0.05 ft. (approximately 19⁄32 in.) is a very definite indication that the strata have been broken or at least deflected downward, and is not due to errors in leveling.

In Table 1 the angles of draw are given for this initial subsidence of 0.05 ft. In protecting a surface structure, the value of the coal to be left in place, the value of the structure to be preserved, and the cost of repairs to a structure that may be partly damaged, are factors which should be considered in addition to the angle of draw. It is of interest to know what the angle of draw would be, if the amount of subsidence is to be confined to the following limits: (1) more than 0.10 and less than 0.20 ft.; and (2) more than 0.20 and less than 0.30 ft. Table 1 gives the angles of draw for these limits also.

All angles of draw are called "positive" when the subsidence was occurring ahead of the rib or extraction line and over the solid coal, and are called "negative" when the line of draw extends back of the extraction line. The latter did not occur in any instance covered by this study. The term "solid coal" here means coal in large pillars, only 25 per cent of the original area having been previously extracted by the headings,

TABLE 1.—*Angles of Draw*

Date, 1932	A Line				B Line			
	Rib Line at Station	Angles of Draw			Rib Line at Station	Angles of Draw		
		0.05′	0.1′ to 0.2′	0.2′ to 0.3′		0.05′	0.1′ to 0.2′	0.2′ to 0.3′
Feb. 8	3 + 02	18°35′	9°51′	0°23′	3 + 20	21°43′	8°17′	−0°57′
Feb. 22	2 + 89	27°55′	11°59′	7°25′	2 + 59	18°50′	1°39′	−2°58′
March 8	2 + 15	20°17′	2°50′	−1°54′	2 + 59	22°45′	8°23′	6°10′
March 21[a]	2 + 15	20°17′	7°28′	2°50′	2 + 30	22°03′	9°47′	3°14′
April 4	2 + 15	24°06′	11°57′	7°28′	2 + 30	25°42′	9°47′	5°12′
April 18	2 + 15	24°06′	16°13′	11°57′	2 + 30	25°42′	14°03′	9°47′
May 2[a]	1 + 50	21°37′	9°08′	4°37′	1 + 55	18°00′	9°44′	0°54′
May 16	1 + 50	25°19′	13°28′	7°20′	1 + 55	21°51′	9°44′	5°22′
June 1	1 + 23	[b]	8°43′	4°14′	1 + 55	21°51′	13°57′	9°44′
June 13	0 + 76		4°44′	0°11′	1 + 17	15°50′	11°43′	7°26′
June 27	0 + 76		4°44′	0°11′	1 + 17	19°47′	11°43′	7°26′
July 11	0 + 76		4°44′	0°11′	1 + 17	20°59′	11°43′	7°26′
July 25	0 + 76		9°11′	0°11′	1 + 17	23°41[b]	11°43′	7°26′
Aug. 8	0 + 76		9°11′	4°44′	1 + 17	23°41[b]	11°43′	7°26′
Aug. 22	0 + 76		9°11′	4°44′	1 + 17	25°01[b]	15°50′	11°43′
Sept. 6	0 + 53		9°30′	0°33′	0 + 47	[b]	3°53′	−0°32′
Sept. 19	0 + 20		[b]	3°37′	0 + 47		8°13′	3°53′
Oct. 3	0 + 20			[b]	0 + 47		[b]	8°13′
Oct. 17	−48				0 + 12			[b]
Oct. 31	−74				0 + 12			
Nov. 14	−114				−23			
	C Line				D Line			
Feb. 8	2 + 79	17°33′	0°43′	−3°46′	3 + 06	17°45′	5°24′	1°03′
Feb. 22	2 + 79	21°18′	5°07′	0°43′	2 + 41	11°09′	−1°34′	−4°00′
March 8	2 + 70	23°36′	12°04′	3°32′	2 + 20	11°46′	−0°52′	−5°12′
March 21[a]	2 + 57	21°41′	9°52′	5°36′	2 + 20	11°46′	3°26′	−2°36′
April 4	2 + 40	19°07′	11°11′	6°58′	2 + 20	11°46′	3°26′	−0°52′
April 18	2 + 00	12°44′	4°21′	0°00′	1 + 90	6°47′	2°34′	−3°52′
May 2[a]	1 + 60	14°18′	6°01′	−2°37′	1 + 75	12°30′	4°15′	0°00′
May 16	1 + 60	18°14′	6°01′	1°44′	1 + 75	12°30′	4°15′	0°00′
June 1	1 + 60	18°14′	10°13′	6°01′	1 + 75	16°26′	8°26′	4°15′
June 13	1 + 60	18°14′	10°13′	6°01′	1 + 75	16°26′	8°26′	6°15′
June 27	1 + 37	18°31′	6°21′	4°08′	1 + 75	20°10′	12°30′	7°06′
July 11	1 + 26	16°48′	8°42′	2°58′	1 + 75	20°10′	12°30′	8°26′
July 25	1 + 26	20°35′	12°49′	4°28′	1 + 75	23°44′	12°30′	8°26′
Aug. 8	1 + 26	20°35′	12°49′	4°28′	1 + 75	27°06′	12°30′	9°54′
Aug. 22	1 + 26	20°35′	12°49	4°28′	1 + 75	27°06′	12°30′	10°44′
Sept. 6	1 + 23	[b]	12°20′	8°12′	1 + 48	23°24′	8°04′	6°07′
Sept. 19	1 + 23		16°20′	10°17′	0 + 91	[b]	2°42′	−1°33′
Oct. 3	0 + 59		5°48′	1°33′	0 + 73		3°52′	−0°20′
Oct. 17	0 + 59		9°58′	5°48′	0 + 73		8°01′	3°52′
Oct. 31	0 + 33		5°37′	1°22′	0 + 73		12°03′	8°01′
Nov. 14	0 + 33		[b]	5°37′	0 + 42		7°00′	2°51′

[a] Closing error of survey exceeded 0.021 foot.

[b] Subsidence beyond 0 + 00 station.

rooms and crosscuts. During the course of this study, there were periods when the rib line did not retreat for many weeks, and during these periods the angles of draw increased in all three limits. In determining the amount of coal necessary to protect a surface structure completely, the maximum angle of draw would be the proper value to use, although it is possible that even the angles recorded do not represent the maximum angle of draw if the extraction line remained stationary for a long period of time. In applying these angles of draw to other situations, consideration should be given to the fact that the surface sloped upward in the direction of the line of retreating face.

No true indication of surface rise preceding the initial subsidence was recorded. Some surveys showed a slight rise extending for several hundred feet, but this is attributed to instrumental errors. If there was a rise ahead of subsidence, it would pass as a wave from station to station with each successive survey as the rib line retreated, but no such occurrence was recorded.

Rates of Subsidence

Fig. 4 shows the surface profiles for the *B* and *C* lines, together with the position of the coal bed, and the amount of subsidence at intervals of approximately three months. For the *B*-line profile the angles of draw for the limits as given in Table 1 are plotted for the respective quarterly dates.

On the *C*-line profile, the initial angles of draw are shown for every survey up to July 25, 1932; after that date the initial draw was beyond the zero station. The angles of draw up to Jan. 25, 1932, are probably not representative, because some subsidence probably occurred up to station 2 + 50 before the study was begun. From March 8 to April 18 the rib line retreated 70 ft., but the initial subsidence remained at station 1 + 25. When the rib line does not move, the angle of initial draw gradually increases, as shown by the surveys from May 2 to July 25. The data of all the surveys show that for a fast-moving rib line the initial angles of draw are smaller and for a slow-moving or stationary rib line the angles of draw are larger. In other words, the rate of retreat may be faster than the rate of adjustment of the overlying strata.

Figs. 5 to 8 show the surface profiles for every survey and the position of the rib line on each survey date for lines *A* to *D*, respectively. The cumulative subsidence for each date is plotted with respect to a common zero datum plane. These profiles are given only to the 4 + 00 stations, because there was movement between the 4 + 00 and 5 + 00 stations prior to Dec. 1, 1931. The rate and amount of subsidence at any station depends upon: (1) the distance from the break line to the station; (2) the time interval before and after the break line passes the station; and (3) the rate of retreat of the extraction, which may also be termed

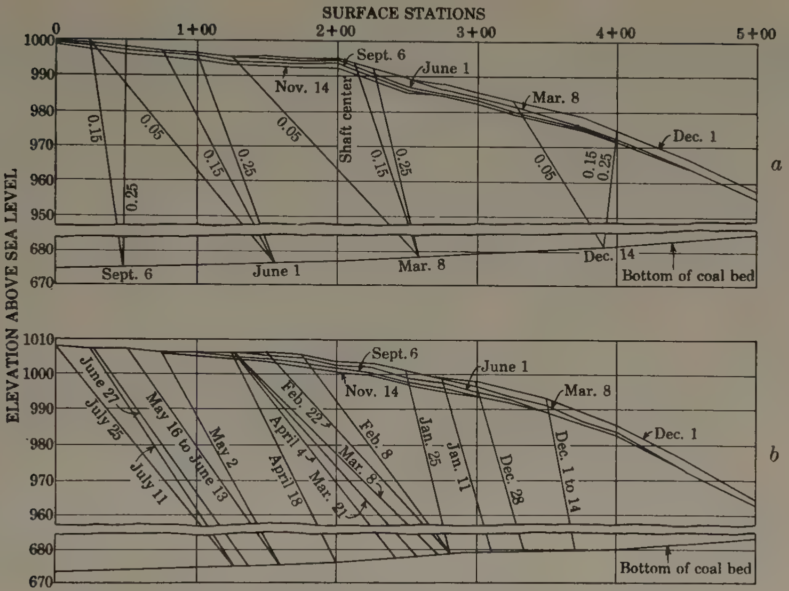

FIG. 4.—PROFILES SHOWING SUBSIDENCE AND ANGLES OF DRAW.
a. *B*-line profile, various limits, Dec. 1, 1931 to Sept. 6, 1932.
b. *C*-line profile, for 0.05 ft. subsidence, Dec. 1, 1931 to July 25, 1932.

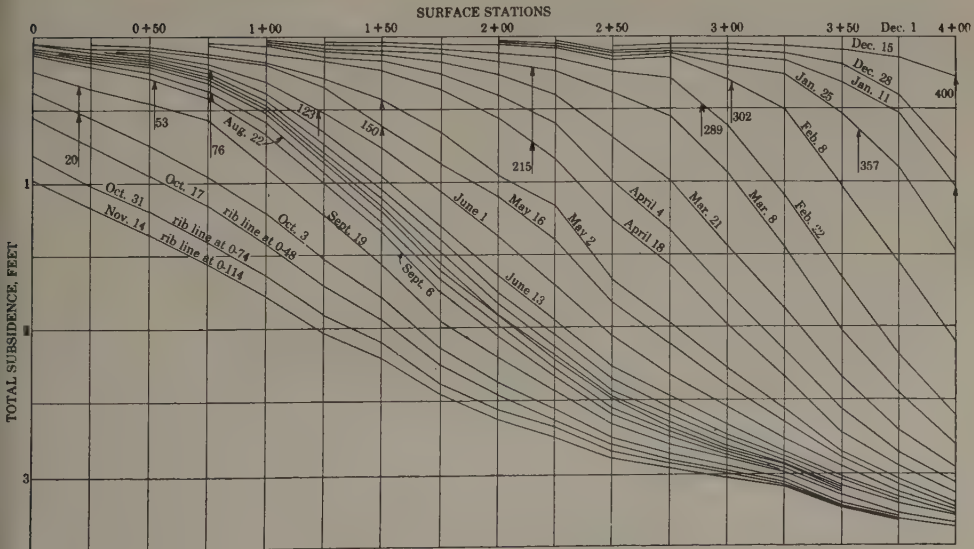

FIG. 5.—*A*-LINE PROFILE SHOWING AMOUNT OF SUBSIDENCE AT EACH STATION FOR EACH SURVEY, DEC. 1, 1931 TO NOV. 14, 1932.
Arrowhead and figure show position of rib line for each survey.

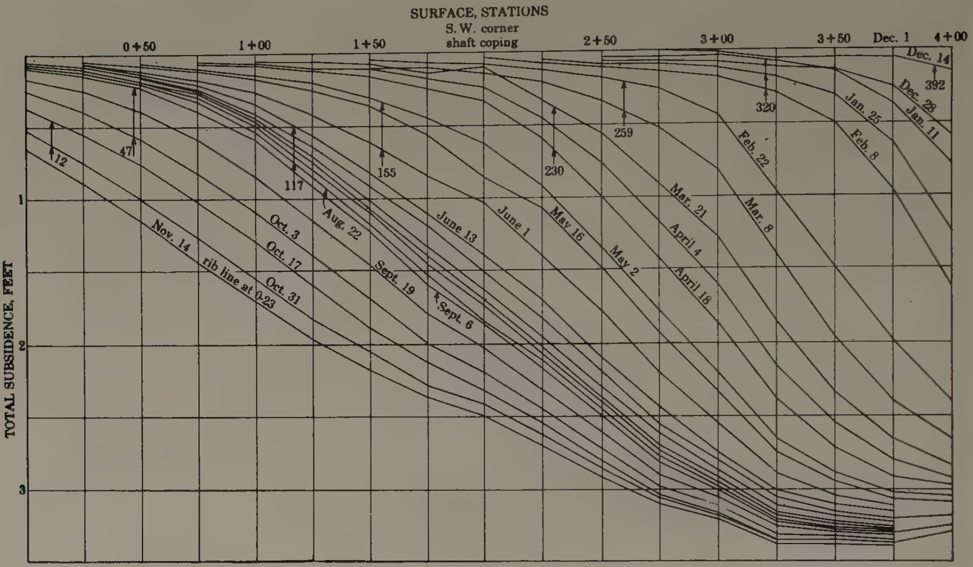

FIG. 6.—*B*-LINE PROFILE SHOWING AMOUNT OF SUBSIDENCE AT EACH STATION FOR EACH SURVEY, DEC. 1, 1931 TO NOV. 14, 1932.
Arrowhead and figure show position of rib line for each survey.

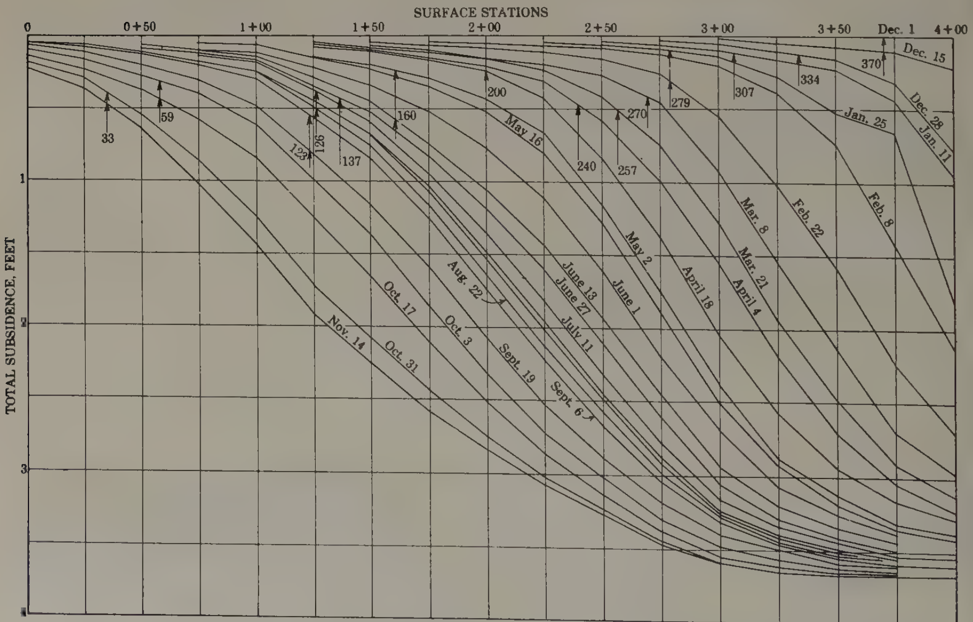

FIG. 7.—*C*-LINE PROFILE SHOWING AMOUNT OF SUBSIDENCE AT EACH STATION FOR EACH SURVEY, DEC. 1, 1931 TO NOV. 14, 1932.
Arrowhead and figure show position of rib line for each survey.

"advance" with respect to unmined coal in pillars. Although not determinable in this case, the rate of subsidence may be affected also by the depth of the workings, the thickness of the coal bed and the amount of waste filling.

The middle curve in Fig. 9*a* is an average subsidence profile based on more than 800 values, showing the relation between total subsidence and distance from station to rib line. The amount of subsidence is plotted for every 10 ft. of distance in front of and behind the rib line. The right end of the curve is irregular, because not enough values could be obtained to give a true representative average for these points. This curve shows the average amount of subsidence to be expected at any point on the surface with respect to its distance from the gob with a uniformly retreating rib line. While total subsidence is plotted, the difference in subsidence between any two points on the curve represents the rate of subsidence to be expected while the rib moves the horizontal distance between the two points. The maximum and minimum values of subsidences are plotted as curves for every 10 ft. in distance. These curves are more irregular than the curve of averages, because only single values are represented for each distance. After the rib line passes a station, the faster it moves the greater is the rate of subsidence at the station; and, the slower it moves, the slower the rate of subsidence at the station. This accounts

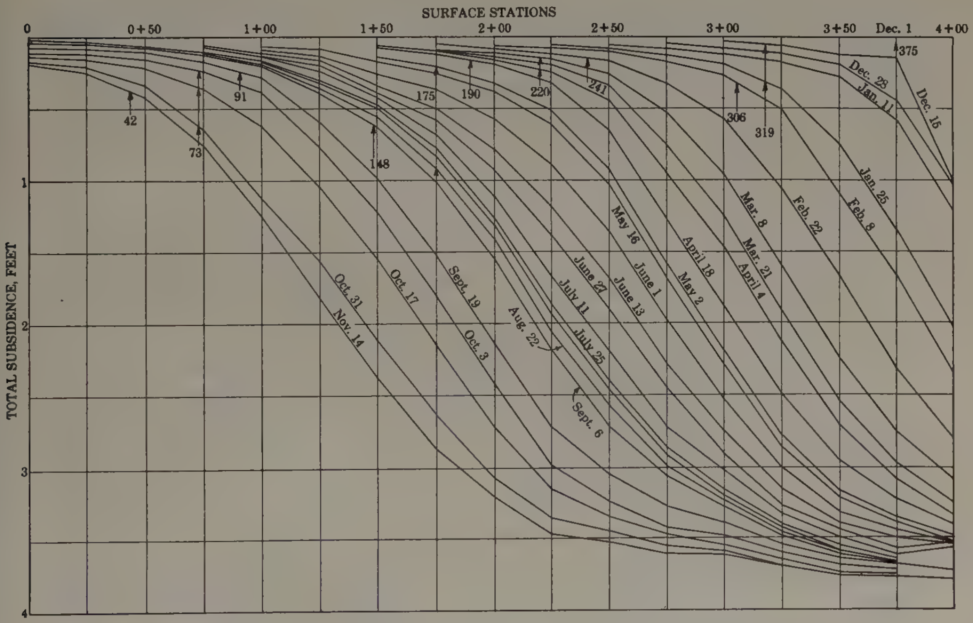

FIG. 8.—*D*-LINE PROFILE SHOWING AMOUNT OF SUBSIDENCE AT EACH STATION FOR EACH SURVEY, DEC. 1, 1931 TO NOV. 14, 1932.
Arrowhead and figure show position of rib line for each survey.

for the high and low points on the maximum and minimum curves. The minimum subsidence curve over solid coal is the result of a fast-moving rib line, and the maximum subsidence curve is due to a slow-moving rib line. The three curves start at the left at a common point at the beginning of appreciable subsidence and if they were carried to the point of complete subsidence would again approach each other.

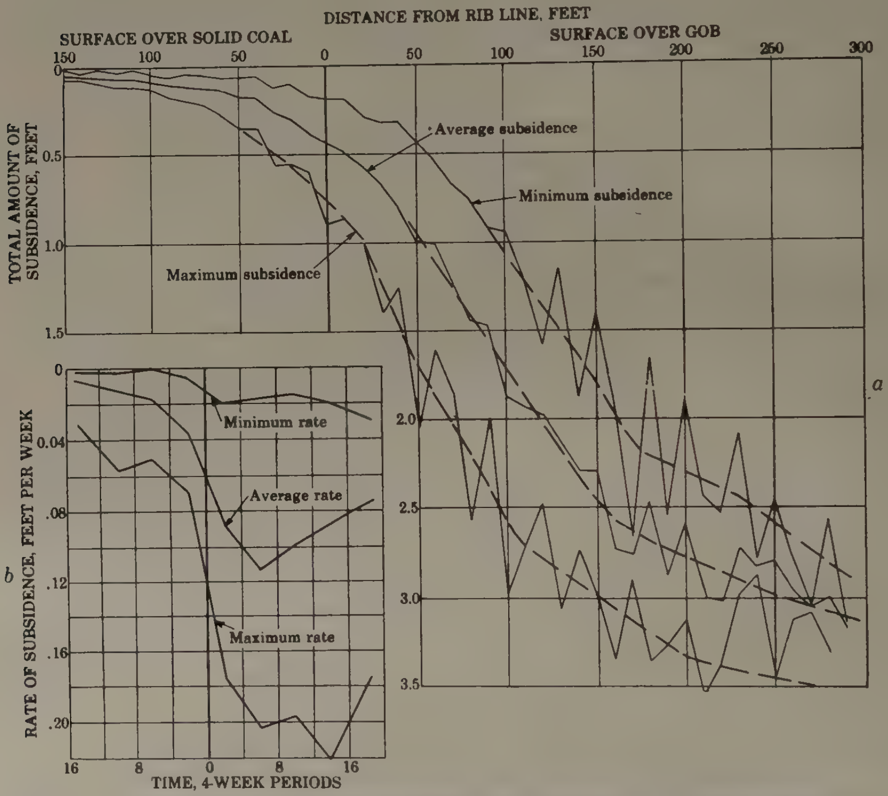

FIG. 9*a*.—AMOUNT OF SURFACE SUBSIDENCE WITH RESPECT TO DISTANCE FROM RIB LINE.

FIG. 9*b*.—RATE OF SUBSIDENCE WITH RESPECT TO TIME INTERVAL BEFORE AND AFTER RIB LINE PASSES A POINT ON SURFACE.

Fig. 9*b* shows the relationship between rate of subsidence and time both before and after the rib line passes a station. The maximum rate of subsidence is reached in the period from 4 to 8 weeks after the rib line has passed the station, but, of course, the total amount of subsidence continues to increase. The maximum and minimum rates are also plotted in Fig. 9*b*. Up to the time that the rib line passes the station, the minimum rate of subsidence is the result of a fast-moving rib line and the maximum rate of subsidence is the result of a slow-moving rib line. After the rib line passes the station, the reverse is true: a slow-moving rib line results in values as shown by the minimum curve, and a fast-moving rib line results in values shown by the maximum curve.

Horizontal Movement

Marked horizontal movement developed as the surface settled. This movement was chiefly longitudinal—parallel to the survey lines—although in all cases subsequent lateral movement developed, but reached a maximum of only 0.27 ft., whereas the longitudinal movement reached a maximum of 1.24 feet.

The average amount of horizontal movement with respect to the three limits of subsidence given in Table 1 (and also indicated in Figs. 10*a* and 10*b*) is as follows: For 0.05 ft. of subsidence, longitudinal movement was 0.03 ft. and lateral movement nil; for 0.15 ft. of subsidence, longitudinal movement was 0.11 ft. and lateral movement 0.02 ft.; and, for 0.25 ft. of subsidence, longitudinal movement was 0.22 ft. and lateral movement 0.04 ft. While a point is sinking from its original position to 0.50 ft., the amount of longitudinal movement is practically equal to the amount of subsidence. The average maximum longitudinal movement reached before a point started to return to its original position was 0.84 ft. All longitudinal movement was first towards the gob or mined-out area to the limits given and then the movement was back towards the original location. The longitudinal movement towards the mined-out area may or may not have been affected by the slope of the surface in that direction, nevertheless after the maximum was reached the affected areas of surface began to move back towards their original location.

Figs. 10 and 11 are graphs representing horizontal movement with respect to subsidence. Since the rib was at station D-1 + 75 from May 2 to Aug. 22, the values for stations D-1 + 00 to D-2 + 50 are used, because they are not only representative but also show the relationships between subsidence, horizontal movement and the horizontal distance from the station to the rib. In Figs. 10 and 11 the distance between points on each curve represents intervals of four weeks. The zero coordinates represent the original position of each station. The numerical values opposite each point represent the distance from the station to the rib, and if no value is given it is the same as the preceding and succeeding marked points. A plus sign indicates that the station is over solid coal and a negative sign that the station is over the gob by the amount indicated from the rib or fracture line. The larger chart of each figure shows the longitudinal movement and subsidence; the smaller chart shows the longitudinal and the lateral movement. The vertical and horizontal scales are equal and therefore there is no graphical distortion.

Fig. 10*a* shows conditions at three stations on *D* line (Fig. 1) at +25, 0 and −25 ft. from the rib line, and it should be noted that for the time that the rib line remained stationary the amount of subsidence at each station was as might be expected; whereas the amount of longitudinal movement at the station over the solid coal was only 0.10 ft. less than

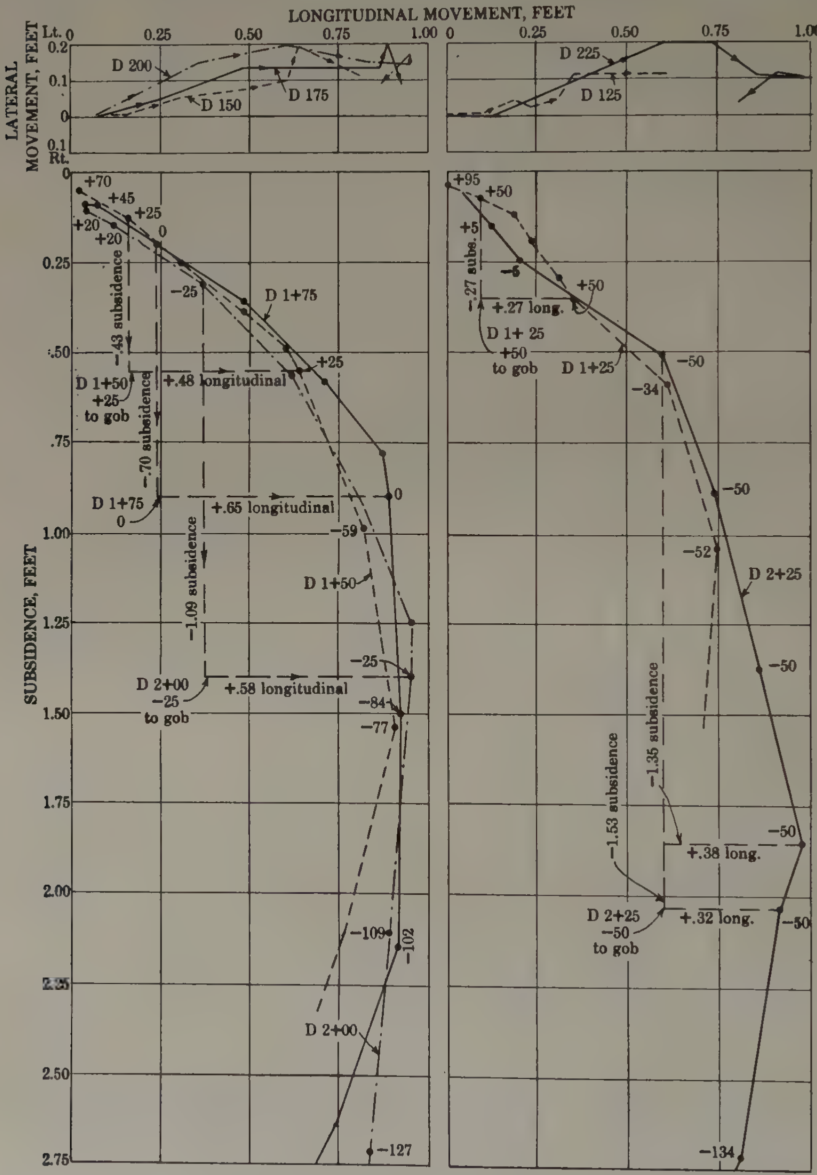

FIG. 10.—HORIZONTAL MOVEMENT WITH RESPECT TO SUBSIDENCE.
a, for stations D-150, 175 and 200.
b, for stations D-125 and 225.

the one over the gob. Fig. 10*b* shows the relationships for points ±50 ft. from the rib line. Here the ground at the station over the solid coal is

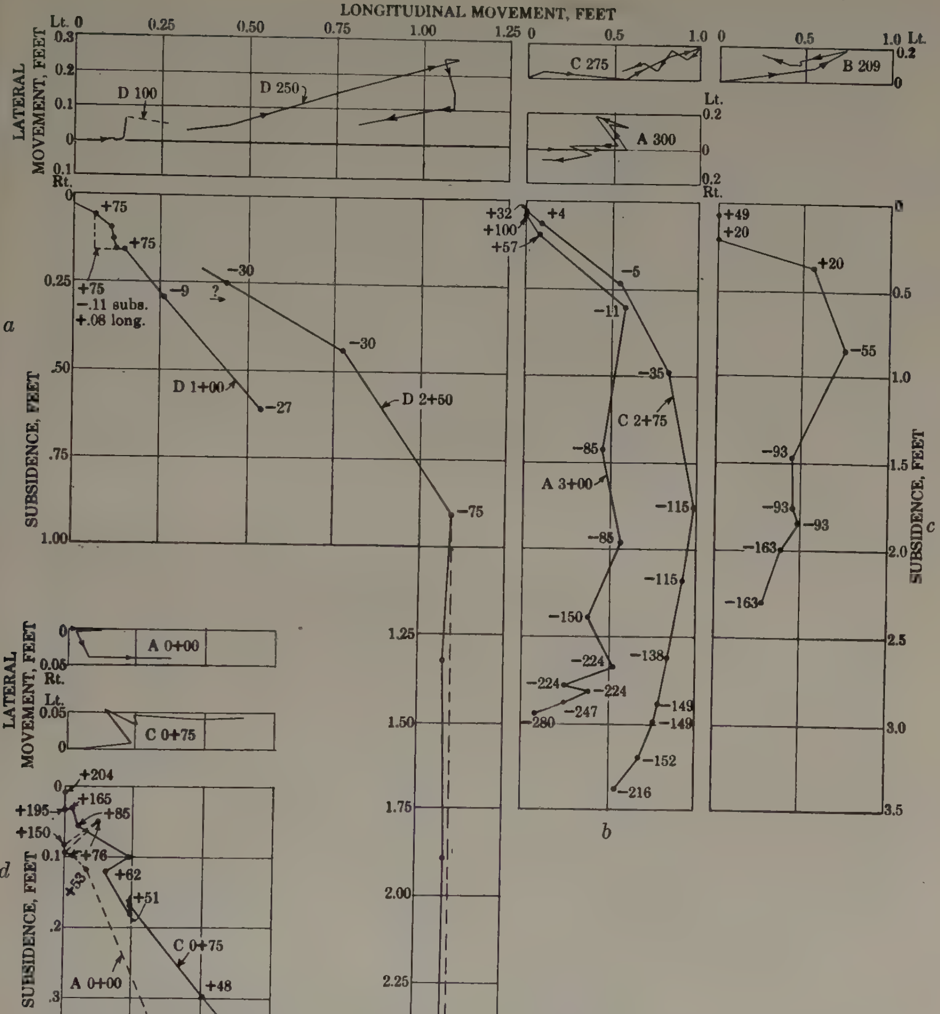
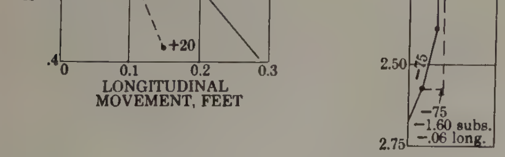

FIG. 11. *abcd*

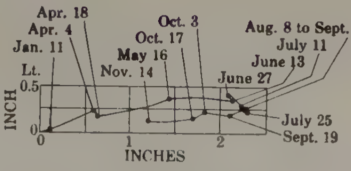

FIG. 12.

FIG. 11.—HORIZONTAL MOVEMENT WITH RESPECT TO SUBSIDENCE.

a, for stations D-100 and D-250.

b, for stations A-300 and C-275.

c, at shaft top, movement based on punch mark on rail at station 209 and subsidence on northwest corner of coping at station 212.

d, for stations A-0 + 00 and C-0 + 75.

FIG. 12.—DIFFERENTIAL MOVEMENT OF RAFTERS AND FLOOR OF SUBSTATION BUILDING AS INDICATED BY PLUMB LINE.

undergoing considerable horizontal pull but not much subsidence, whereas that at the station over the gob is not having a great amount of longitudinal movement but is rapidly subsiding. Fig. 11*a* shows the condition for points ±75 ft. from the rib line. In both cases the amount of horizontal movement is small, and, as might be expected, the station over the solid coal has subsided only a small amount, whereas the one over the gob has subsided considerably.

The graphs of horizontal movement in relation to subsidence and position of rib line of the *D* stations are representative of all the stations but, to complete the picture, a few other representative stations are given. Fig. 11*c* shows the movement at the shaft top, which shows that horizontal movement did not begin until the rib line was within 30 ft. of the shaft center. Fig. 11*b* shows the movement for two stations where the rib line retreated regularly. Fig. 11*d* shows the small movements that occur while the rib is still at a considerable distance from the station.

Despite the rather marked horizontal movements, only a few surface breaks occurred. These developed on the relatively steeper hillside slopes at the 500 stations and were parallel with the rib line. Manifestly they were due to ground slippage near the surface. The most marked surface break was a crack 2 to 3 ft. wide and 200 ft. long, in which it was possible to look down 20 or 30 ft. As stated, there were a few other breaks, but none approached this in magnitude, indicating that in flat country subsidence may occur unnoticed.

A plumb line 14 ft. long was hung from the rafters of the substation building, and every two weeks the differential horizontal movement of the plumb-bob point was plotted (Fig. 12). It is of the same nature as all the other horizontal movements but not of the same magnitude, because the floor of the substation building was moving as well as the rafters. However, the graph shows the effect of subsidence by a slight tilting of the structure towards the gob.

On May 31, 1934, measurements of longitudinal movements were made along the *B* and *C* lines. Twenty representative stations, which had moved an average of 0.84 ft. longitudinally, had moved back until they were only an average distance of 0.32 ft. from their original positions.

Shaft Movement

A series of shaft measurements was made up to and including March 1, 1932. After that date no further measurements could be made because it was necessary to remove the electric-power lines from the shaft, and no other source of power was available to operate the hoist. Shortly after this date it was necessary to seal the top of the shaft to control the ventilation along the rib line. Therefore the observations are limited, particularly since the initial angle of draw did not pass through the shaft until Feb. 8. Table 2 shows the amount of horizontal movement of the walls of the shaft from their original position to their positions on Feb. 8, Feb. 22, and March 1, 1932, corrected for movement of the plumb lines. It should be stated that the original position of the walls was determined by averaging the measurements of the first three surveys, and the values in Table 2 are based on the changes from this average. It is estimated that the possible error in measurement was ±0.04 ft., and only values greater than 0.04 ft. are given for the three surveys listed. All measurements of surveys prior to Feb. 8 were less than the possible error. The value 0.04 was used because the average of the maximum variations of the first three surveys at the 62 shaft stations was 0.041 foot.

Table 2.—*Horizontal Movement of Shaft Walls in Decimal Parts of a Foot*

Vertical Distance below Shaft Coping, Ft.	Movement of Wall, Decimal Part of One Foot					
	At Station 1 + 96[a]			At Station 2 + 04[b]		
	Feb. 8	Feb. 22	March 1	Feb. 8	Feb. 22	March 1
00	0.11	0.14	0.16		0.10	0.11
10	0.06	0.13	0.14		0.05	0.09
20	0.07	0.11	0.11		0.04	0.07
30	0.06	0.09	0.09		0.06	0.09
40		0.08	0.07		0.07	0.09
50		0.08	0.07		0.02	0.11
60		0.05			0.06	0.05
Rib at				3 + 20	2 + 59	2 + 59
Initial surface subsidence at B station				1 + 93	1 + 50	1 + 25

[a] Southeast wall of shaft (farthest from approaching retreat line).
[b] Southwest wall of shaft (nearest to approaching retreat line).

All lateral movements were in the direction of the gob or goaf. The values given represent the movement of the shaft walls, based on measurements from the plumb lines and after correcting for the movement of the plumb lines.

From these data, Fig. 13 was prepared. Since the shaft measurements are measurements of horizontal movement, some idea of the amount of subsidence represented was necessary. This was obtained by averaging all the longitudinal movements and subsidence of the surface monuments, and it was found that an average longitudinal movement of 0.04 ft. equaled an average subsidence of 0.06 ft., and an average longitudinal movement of 0.16 ft. equaled an average subsidence of 0.19 ft. Values between these limits were in proportion. Fig. 13 shows that the line of break to the surface is not necessarily a straight line, but is more probably a curved line concave to the coal bed. The line of Feb. 8 is shown straight, because the initial subsidence at the surface was close to the shaft and the concrete was tending to reinforce the natural rock structure at this shallow depth. However, on Feb. 22 and March 1, the point of initial movement of the shaft is much higher above the coal than can be expected if the break is a straight line. The angle of draw for 0.15 ft. of subsidence did not pass through the shaft until March 21, and nothing can be stated with respect to it, or to 0.25-ft. subsidence.

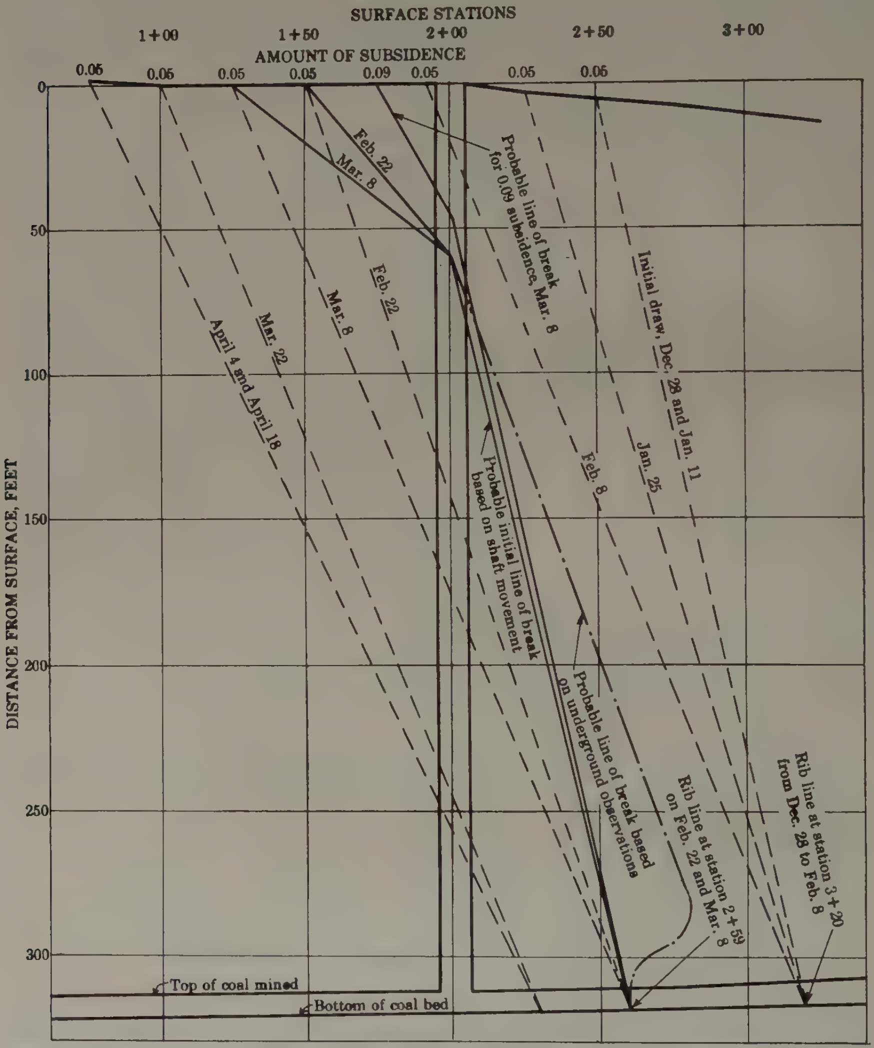
SURFACE STATIONS
1+00
1+50
2+00
2+50
3+00
AMOUNT OF SUBSIDENCE
0.05
0.05
0.05
0.05
0.09
0.05
0.05
0.05
0
50
100
150
200
250
300
DISTANCE FROM SURFACE, FEET
Feb. 22
Mar. 8
Probable line of break for 0.09 subsidence, Mar. 8
Feb. 22
Mar. 8
Mar. 22
April 4 and April 18
Initial draw, Dec. 28 and Jan. 11
Jan. 25
Feb. 8
Probable initial line of break based on shaft movement
Probable line of break based on underground observations
Rib line at station 2+59 on Feb. 22 and Mar. 8
Rib line at station 3+20 from Dec. 28 to Feb. 8
Top of coal mined
Bottom of coal bed

FIG. 13.—LINE OF BREAK OF ROCK STRATA BASED ON SHAFT MEASUREMENTS, SURFACE SURVEYS AND UNDERGROUND OBSERVATIONS.

However, a line is shown on Fig. 13 for the line of break of 0.09-ft. subsidence on March 8. It is unfortunate that it was not possible to continue the shaft readings after March 1 so that additional data could have been obtained to confirm further the position of the break lines as shown. On March 1 there was 59 ft. of solid coal from the shaft center to the rib line, and the movement developed in the shaft, although measurable, had not materially affected the soundness of the shaft.

Fig. 14 illustrates the layout at the shaft surface with the points on which elevations were taken every two weeks. At the time the shaft was covered the southeast and northeast corner monuments were lost. Fig. 14 shows a profile of the northwest and southwest corners of the shaft coping, a bolt bench mark and stations B-1 + 75 and B-2 + 25. On March 21 the rib line was at station 2 + 30, and the average total subsidence at the three points on the shaft coping amounted to 0.15 ft. On

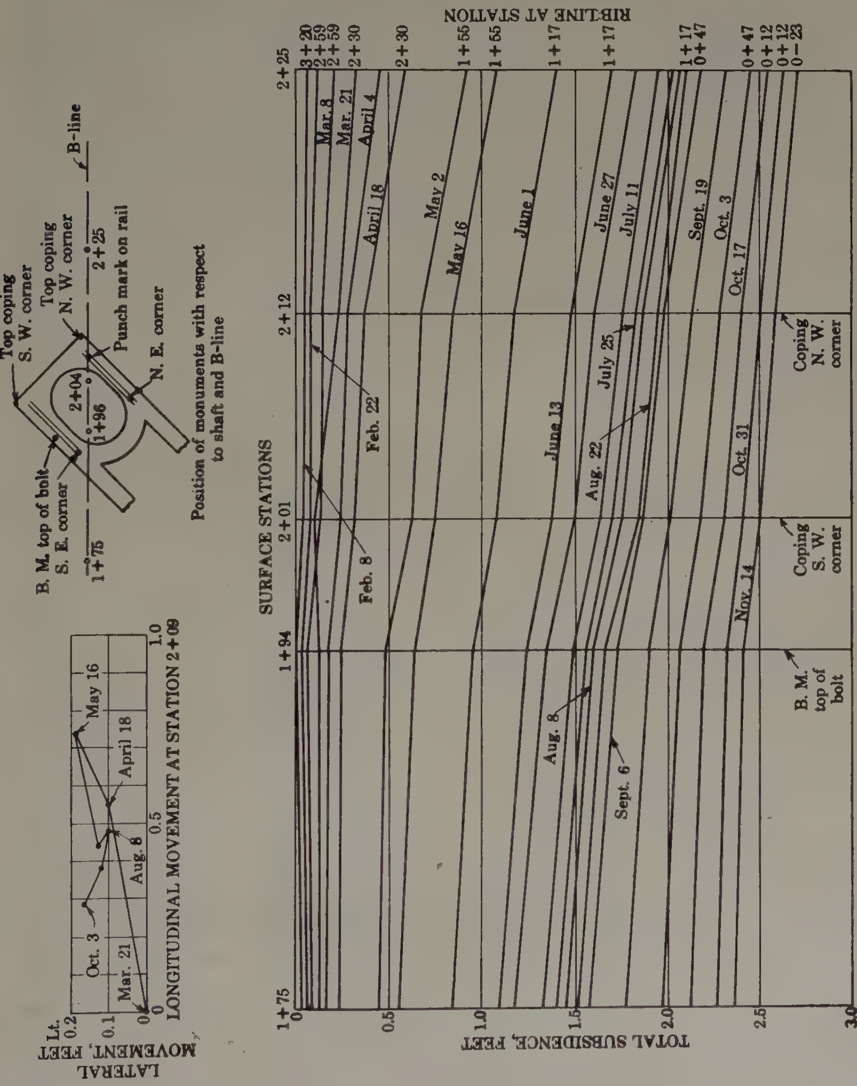

FIG. 14.—DETAILS OF SUBSIDENCE AT SHAFT TOP DURING 1932.

April 18 the rib line was still at station 2 + 30, and the average total subsidence had increased to 0.31 ft. In addition, the punch mark on the rail moved longitudinally a distance of 0.55 ft. and laterally 0.10 ft. in the period from March 21 to April 18. While no observations were possible in the shaft during this period, these values indicate that when the rib line was 30 ft. from the shaft center the shaft was damaged, at least close to the surface. While the rib line was at station 2 + 59, the average amount of total subsidence was 0.11 ft., and there had been no horizontal movement other than that measured by the plumb lines. It is probable that, if the rib line had remained at station 2 + 59, the angle of draw would have increased sufficiently to affect the shaft. The angle from the rib at 2 + 59 to the surface at station 1 + 90 is 12° 13′ and this would indicate a subsidence of at least 0.15 to 0.20 ft. The effect on the shaft lining, therefore, would be dependent upon the strength of the lining to resist shear by adjacent ground movement.

It is probable that for a small amount of subsidence the concrete lining of the shaft tended to strengthen the adjacent ground. Table 2 shows that the wall farthest from the gob moved slightly more than the wall closest to the gob. Probably this is due to the fact that the enclosing strata in the former case pressed against and caused movement of the wall; in the latter case, the strata simply drew away from the concrete wall.

On April 12, 1933, the shaft cover was removed and an inspection of the shaft was made from the surface by George S. Rice, Chairman of the Committee on Ground Movement and Subsidence, and the authors. Mirrors were used to reflect sunlight down the shaft. On this date the break line was 450 ft. beyond the shaft and the shaft coping had subsided an average of 2.7 ft. The observations made by Mr. Rice and concurred in by the authors were:

1. The shaft lining was in substantial condition except for a horizontal movement of about 4 in. along a shear plane about 40 or 50 ft. from the top of the shaft.
2. There was about 20 ft. of fill at the bottom of the shaft due to concrete spalling but principally due to crushing at the shaft bottom and caving of the roof in the immediate vicinity of the shaft bottom as the coal was removed.
3. By dropping pebbles along the sides of the shaft walls it was estimated that the shaft was out of plumb by at least 5 in. but not more than 8 in., the inclination being towards the areas first mined. To summarize, the shaft is apparently intact but has suffered both subsidence and horizontal movement, which has caused horizontal breaks in the shaft wall and some scaling of the concrete but has not dislodged any large masses. The strata around the shaft lining have moved but the strength of the shaft walls was great enough to withstand these forces without great damage.

The shaft opening will remain for many years, as the maximum damage possible by subsidence has already occurred. In its present condition, with slight repairs it could be used for working a deeper coal bed if such exists.

On May 31, 1934, the average subsidence for stations B-1 + 75 and B-2 + 25 was 2.80 ft. and the average subsidence for three points on the shaft coping was 2.73 feet.

OBSERVATIONS ON HIGH-SCHOOL BUILDING

In another part of the same mine, about ½ mile southeast of the air shaft, mining operations were being carried on under a school building.

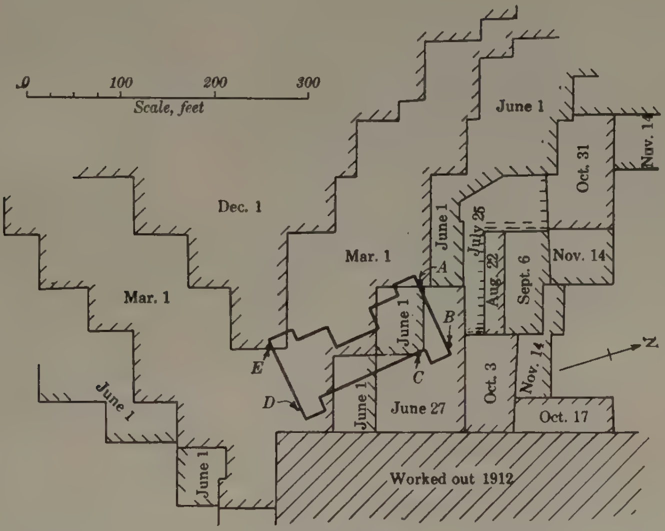

FIG. 15*a*.—POSITION OF RIB LINES WITH RESPECT TO HIGH-SCHOOL BUILDING.

A series of observations, to obtain data of subsidence effects under complicated conditions of rib extraction, were made as the mining progressed. Fig. 15*a* shows the plan of the school building and the rib lines on various dates. The building was constructed with brick walls and timber floor beams. The rib line was retreating along an old worked-out area and the mine development made it necessary to form two rib lines, the first to continue its retreat to the north and the second to retreat to the east. As shown on Fig. 15*a*, this change was made under the south end of the building.

The change in elevations of all the points cannot be shown clearly on a profile, but by using as coordinates the subsidence in feet and the time interval in days, it is possible to show the changes that took place at five points near the base of the building. The locations are shown as *A*, *B*, *C*, *D* and *E* on Fig. 15*a*. It is probable that at point *E* some movement had occurred before the first reading. Fig. 15*b* shows the changes at the five points. Points *D* and *E* subsided from the beginning, bad vertical cracks opening at the southwest corner. Points *A* and *C* acted more or less as fulcrums; and point *B* rose 0.39 ft. before it began to settle. From Fig. 15*a* it is evident that during the period March 1 to June 1, 1932 very little mining was done, but that during this period the south end of the building was over gob; whereas the north end of the building was over solid coal. The rise shown at point *B* may be due to tilting in the underlying rock strata but may be due also to the fact that the foundations of the building acted as a weak beam. The building rests on the Waynesburg sandstone, which is about 60 ft. thick. The depth of the coal bed at the school is 402 ft. The south end of the building was badly cracked, and it was necessary for the school board to fence off and not use the classrooms in that end.

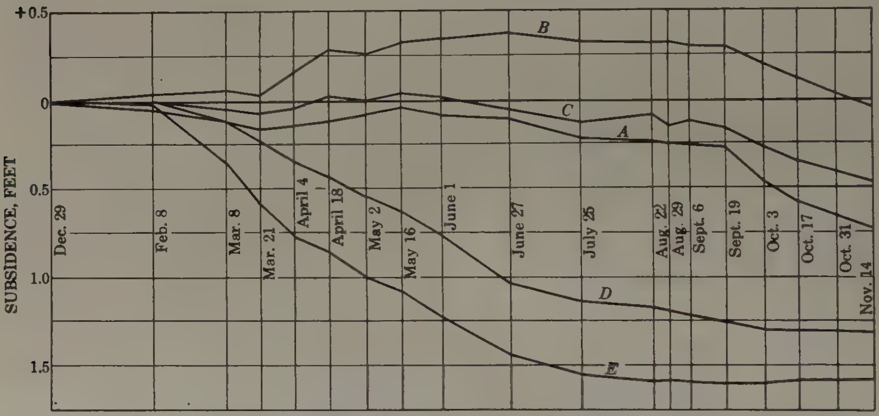

FIG. 15*b*.—SUBSIDENCE TIME CURVE OF FIVE POINTS ON HIGH-SCHOOL BUILDING, DEC. 29, 1931 TO NOV. 14, 1932.

A survey on May 31, 1934 indicates that the building is returning to a more nearly level condition. Point *B*, which at one time rose 0.39 ft., has now sunk to 0.32 ft. below its original position. Point *E*, which had the maximum amount of subsidence of 1.60 ft., has risen until now it is 1.04 ft. below its original elevation. Points *B* and *E* on June 27, 1932, had a difference in elevation of 22 in. but on May 31, 1934, this difference amounted to only 9 in. An expenditure of $3000 has put the building in good condition.

GENERAL CONCLUSIONS

As regards the angle of draw, the first obvious conclusion is that the ultimate angle of draw cannot be ascertained, when a rib line is rapidly retreating, because, as shown, the overlying rocks do not necessarily readjust themselves to their new stresses as rapidly as the rib line may retreat. In only one case out of 11 instances, where the rib line remained at the same station for two or more surveys, did the initial angle of draw remain the same; otherwise the angle increased from 3° 39′ to as much as 14° 36′, or an average increase for all 11 instances of 4° 45′. For subsidence to the amount of 0.15 ft., the angle averaged 5° 36′ for 12 instances, and for subsidence of 0.25 ft. the average in 14 instances was 6° 41′.

Closely spaced monuments and surveys at frequent intervals are necessary to measure the surface movement properly. For instance, if surveys had been made at three-month (12-week) intervals on March 21, June 13 and Sept. 6, the angles of draw measured for the three amounts of subsidence would have been 23° 24′, 11° 43′, and 8° 12′ instead of the maximum values determined by this work as 27° 55′, 16° 20′, and 11° 57′. This is an average difference of 4° 18′, which, with 320 ft. of cover, roughly indicates a difference of 25 to 30 ft. in the amount of coal necessary to protect a surface structure, depending upon the amount of subsidence considered permissible. If surveys had been made every three months on stations 100 ft. apart, the indicated results would have been very erratic.

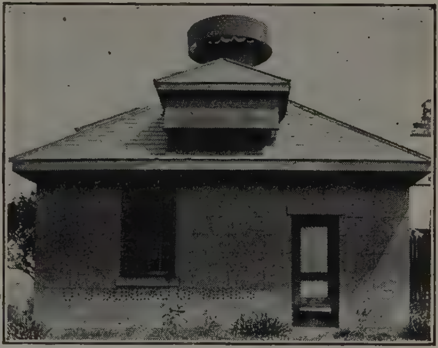

FIG. 16.—SOUTH SIDE OF SUBSTATION BUILDING, MAY 20, 1932.

If a certain amount of surface subsidence is considered permissible for the particular conditions, the amount depending upon the character of the structure to be protected and the value of the coal to be recovered, it should be possible, by good engineering judgment, to recover a greater tonnage of coal and thus reduce the cost of maintaining the structure. Under natural conditions similar to those described in this paper, assuming a structure occupying an area 50 ft. square with 300 ft. of cover and using the maximum angles of draw observed for the various subsidences, which are suggested as not likely to cause serious damage, the area of the coal necessary for protection would have to be in the form of a square, 368 ft. on each side for no subsidence, 226 ft. square for 0.15-ft. subsidence, and 176 ft. square for 0.25-ft. subsidence. If the bed contains 7 ft. of minable coal and averages 24. 5 cu. ft. per ton, by allowing 0.15 ft. of subsidence it is possible to mine an additional 24,100 tons as compared to complete protection; or, if 0.25 ft. of subsidence is permissible, 29,800 tons additional can be mined as compared to complete protection.

The horizontal movement, which results when subsidence occurs, may be more destructive to surface structures than the mere sinking of the ground. However, if complete mining is necessary, it is not possible to control the horizontal movement. Based on longitudinal measurements, it appeared as though stations 25 to 50 ft. apart moved as a unit for a time, but that later differential movement occurred within the 50-ft. limit. There were a few instances, however, where stations 25 ft. apart appeared to move as a unit throughout the period of observation; therefore it is possible that a structure of this dimension may be little affected by surface movement, but also it is probable that the differential movement may occur within the limits of the structure. Figs. 16, 17 and 18 show the effect of subsidence and its accompanying horizontal movement on three structures. Fig. 16 is one side of the substation, which is 22 ft. long and at an angle of 45° to the line of retreat. This photograph was taken on May 20, 1932, and there are no changes evident by comparison with a photograph taken on Dec. 1, 1931. The crack over the door was present on that date. Fig. 17 shows part of the wall around the shaft, which is 37 ft. long and is at 45° to the line of retreat, but at 90° with respect to the wall shown in Fig. 16. The only change recorded in this photograph as compared to a similar one taken on Dec. 1 is the crack marked *BB*. On May 20, the rib line was past the structures shown in Figs. 16 and 17 by 75 and 35 ft., respectively. The average amount of subsidence at near-by monuments was 0.85 ft. Observations in October, 1932, showed no changes other than those given in the photographs. Fig. 18 shows the southwest corner of the high-school building on May 20. On that date the corner of the building was over the gob, whereas about 60 per cent of the building was over unmined pillars of coal. The building is 175 ft. long and 100 ft. wide.

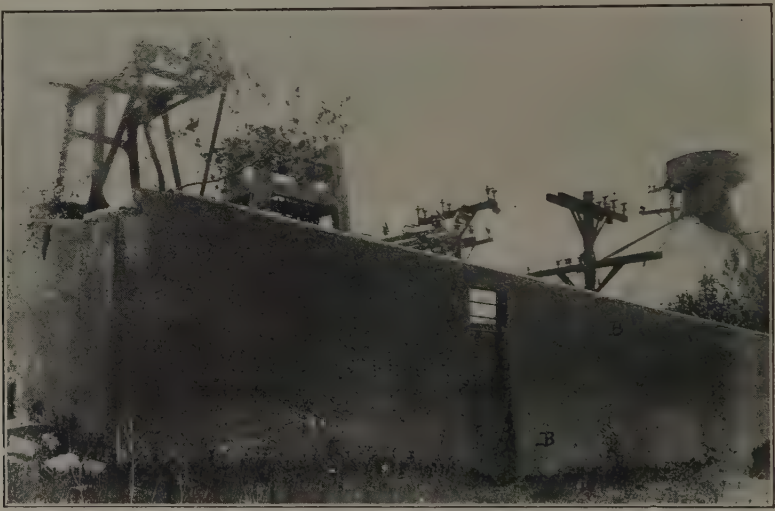

Fig. 17.—West side of wall around air shaft, May 20, 1932.

As indicated by Fig. 13, the plane of actual break does not coincide with the theoretical line of draw; the nature of the break is not a plane surface but is made up of complicated curves, varying with the strength of the different strata. As observed in the mine at the rib line, the break starting from the bottom of the coal bed is vertical up through the coal bed, the draw slate, and a few feet of the overlying shales, and then inclines backwards over the gob at rather large angles with the vertical. Apparently the immediate effect is due to the successive breaking off of horizontal layers of rock, which act like cantilevers anchored over the solid coal and extending over the gob. How far back these overhangs hold up it is difficult to say, but as much as 20 ft. of overhang has been observed in the district. The amount of overhang is probably a function of the rate of retreat and the character of the rocks in the first 50 ft. above the coal bed. Should the rib line stop for a considerable period of time, this overhang would break off and tend to become a vertical shear plane, depending upon the nature of the overlying rock.

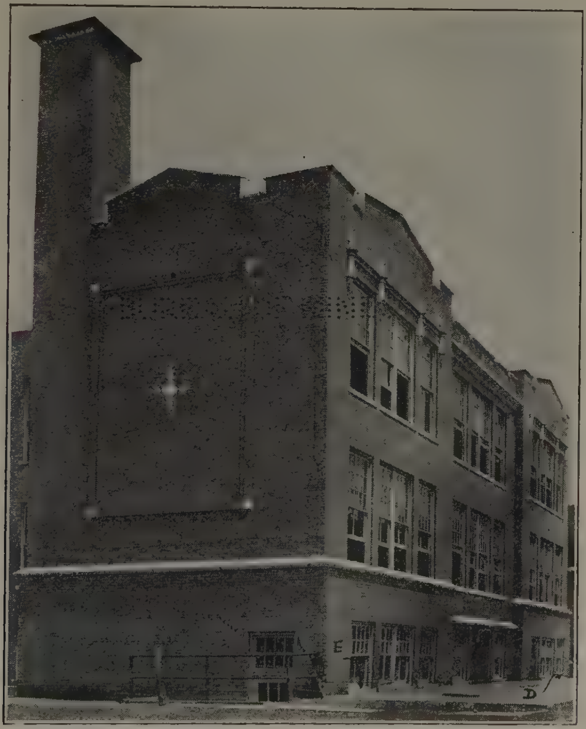

FIG. 18.—SOUTHWEST CORNER OF HIGH-SCHOOL BUILDING, MAY 20, 1932.

Somewhere back over the gob, and probably not at a very great height, the mass of overlying rock is too great for the natural cantilever of rock to support, and complete rock failure results. From this point to the surface, the rocks may assume their natural angle of slope, and for a depth of over 300 ft. this natural slope works back and appears at the surface over the solid coal. What the cover would have to be, under natural conditions similar to those at Merrittstown, so that the initial draw at the surface would be over the gob and not over the solid coal, would be interesting to know.

Surveys made on May 31, 1934 indicate that the maximum subsidence measured was 3.90 ft., or 55.1 per cent of the total thickness of the coal bed mined. It is not supposed that this value represents the total subsidence, as surveys for another year will continue to show slow subsidence, but it is probable that it will not exceed about 60 per cent.

Any study of subsidence at a particular locality is a special case, and from it no universal conclusions can be drawn. Many variable factors influence the rate and amount of subsidence. Besides the natural variables, such as character of rock in all its meanings, depth of cover and surface slopes, there are the man-made variables, such as method and rate of mining and completeness of extraction. Therefore, any conclusions that are drawn based on such a study can only be considered in the light of all these variable factors; but if based on reliable data, conclusions may be deduced which will be of value in planning future workings under similar conditions to lessen or avoid damage to surface improvements.

DISCUSSION

(*George S. Rice presiding*)

H. Rhodes,* Rotherham, England (written discussion).—We rarely have an opportunity nowadays in Great Britain to work such a thick seam as the Pittsburgh bed, or of observing the effect upon the surface of working such a seam at comparatively shallow depths. The results given in this paper, however, largely confirm my experience of the effect upon the surface of working a thinner seam, but at a considerably greater depth than referred to in the paper.

A paper recently contributed to the Institution of Mining Engineers[1] describes the mining of a seam 49 in. thick. The observations extended over a period of about 10 years. In 1920, a preliminary leveling was made to determine the position of the tunnel in relation to ordnance datum before the effect of mining began, and all subsequent levelings were tied on to the original starting point. Leveling stations inside the tunnel, consisting of numbered plates of cast iron spiked to the wall, were one chain apart. Levelings were taken on these plates at about monthly intervals, and also at one-chain intervals for some distance from each end of the tunnel, stout pegs driven into the ground being used as stations.

The tunnel accommodates a single line of rails and is brick-lined, its sectional area being 16 ft. high to the crown, 12 ft. wide at the bottom, and 14 ft. wide at the springing of the arch 8 ft. from the bottom; it is not inverted. The tunnel is about 1000 ft. long, and the cover over it varies from 21 to 57 ft. The strata through which the tunnel is driven consist of coal measures with a thin seam of coal. Preparatory to working the coal, the tunnel arching was reinforced with old steel rails.

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* C. E. Rhodes & Sons.

[1] R. H. Horsley and H. Rhodes: Observations on the Working of a Seam of Coal under a Railway Tunnel. *Trans.* Inst. Min. Engrs. (1933–1934) **87,** 129.

The seam is known as the High Hazel, and the depth below the tunnel varies from 613 to 701 ft., the average inclination of the seam under the tunnel being approximately 1 in 14. The seam is 49 in. thick, and is overlain by a soft shale, known locally as "clod," which varies in thickness from 5 to 12 in. Above this comes blue shale with ironstone bands which, according to the record of the shaft sinking some ¾ mile away, is about 27 ft. thick. The shaft record then shows about 12 ft. of sandstone, and from here to the surface the strata consist mainly of alternating bands of argillaceous and arenaceous shales and thin coal seams.

The method of working was by advancing longwall, with pack gates 30 yd. apart. Cross gates were not packed in as the gates advanced, but were scoured through the goaf about 160 yd. back from the face. The clod was removed from the top of the seam by hand picks, and the clod, the ripping dirt from the ordinary gates and the debris from the scouring completely filled the goaf. All back timber was extracted and a continuous pack wall was run across the face about every 15 ft. During the period of observation the faces moved at the rate of about 240 ft. per annum. The process of scouring showed the goaf to be compressed to a height of 16 to 18 in. at about 480 ft. away from the face. No stoppage of traffic occurred during the working of the coal and maintenance work in the tunnel was negligible.

The rate of subsidence was steady, but the amount was not regular throughout the length of the tunnel, a greater fall being registered about the middle. The average subsidence of all the stations in the tunnel was 68 per cent of the thickness of coal removed, or 56 per cent of the thickness of the strata mined.

Until the workings had proceeded under the tunnel about 700 ft. of a total length of 1000 ft., no draw was observed, but at this point a downward movement was revealed about 70 ft. in advance of the faces. Draw was apparent onwards from this point at every leveling station. The angle of draw when first observed was approximately 5°. Our records showed that about 95 per cent of the subsidence at any point occurred within the first 3½ years of the coal being extracted. There was then a quiescent period of about 3½ years with little or no downward movement, a further slight downward movement during the next 9 months, with no movement for the succeeding 3 years.

Notwithstanding the fact that the goaf was completely packed, the subsidence of 68 per cent would appear to be greater than that recorded by Messrs. Newhall and Plein, where, in the Pittsburgh seam, apparently no packing whatever was carried out. This may be due to the influence of the old workings in a seam, also about 4 ft. thick, lying 265 ft. below the High Hazel, but as this lower seam was worked 25 years before the upper, I should have imagined any influence of the former would long ago have ceased.

I also had the opportunity about the years 1908 to 1910 of observing the effect of working a seam 48 in. thick, lying at an inclination of 1 in 16, some 1300 ft. below a two-track, standard-gage, brick-lined and inverted railway tunnel, about 200 ft. long. In this case, also, the seam was worked longwall, and the extraction of the coal caused no apparent damage. I have no record of the amount of subsidence, but it must have been considerable, as, in addition to the seam referred to, a seam 90 in. thick had been worked by the bord and pillar system at a depth of 500 ft. some 30 years previously and 10 years before the tunnel was driven.

H. Briggs,* Edinburgh, Scotland (written discussion).—I find this paper of especial interest since, not long ago, and in collaboration with Dr. W. Ferguson, I undertook a similar investigation at a small shallow mine at Armadale, West Lothian[2]; and, in view of the wide differences between the circumstances, it is, I think, important to recognize that, on the whole, the results bear a strong resemblance. At Barbauchlaw mine the depth is 140 ft. while that at Merrittstown averages about 318 ft.; at the former the coal is worked by advancing longwall, and at the latter the room-and-pillar system is adopted and pillars are being extracted on a retreating line. There are old workings at higher levels at Barbauchlaw—a factor that always increases both the amplitude of subsidence and the extent of draw—while at Merrittstown, I gather, there is none. In neither instance was any pre-elevation of the ground noticed, and in both--as in all others when the matter has received attention—the time-subsidence curves are of the same general character. In both cases the draw was positive and considerable in amount. In both, again, vertical movements were associated with horizontal oscillations of noticeable magnitude.

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* Professor, University of Edinburgh.

I was, I believe, the first to point out that no observation of draw can be of much value unless the degree of accuracy of the leveling is stated. The present authors have driven home that lesson in excellent style. Table 1 throws an almost painful light on the old slap-dash procedure in which draw was decided from levelings of unspecified or unknown precision, or even from visual indications of damage unsupported by any leveling at all. No wonder the literature is full of baffling discrepancies, and that the American literature in particular contains so many instances of alleged negative draw.

The Institute's Ground Movement and Subsidence Committee would be doing work of fundamental importance if it were to give its ruling as to the minimum vertical movement that is to be heeded in inquiries of this character. At Barbauchlaw we decided to disregard any such movement that was less than 0.02 ft. We considered draw as extending as far as the line at which the surface fall, proved by at least three measurements at monthly intervals, was 0.02 ft. or more. Messrs. Newhall and Plein accept the coarser criterion of 0.05 ft. and are seemingly influenced in taking that minimum by the fact that it accompanies the first measurable indication of horizontal movement. We found that vertical fall both started earlier and finished later than the horizontal displacements. In any event, it is simpler and probably more satisfactory to decide the draw from consideration of vertical movement only. Two minima, namely, 0.02 and 0.05 ft., are, then, already offered. I think 0.05 too large; I suggest that the minimum fall should be less than is likely to cause appreciable damage to a building; also that 0.02 is not too exacting, since properly checked levelings with ordinary engineers' instruments have no difficulty in detecting that movement with reasonable certainty.

Newhall and Plein give 0.03 ft. as (on an average) the limit of reliability for horizontal movement, and with this I have no quarrel at all. If the Committee, having considered the matter from every angle, were to decide upon the same minimum of 0.03 ft. (0.36 in., or 9 mm. very nearly) for both vertical and horizontal disturbances I, for one, would welcome their finding. May I also submit that the minimum value be accepted only when substantiated by the next levels at least? For example, if leveling at regular intervals of time gave the following falls for a certain monument:

(1)	(2)	(3)	(4)	(5)	(6)	(7)	
0.00	0.01	0.03	0.03	0.05	0.08	0.11	etc.,

draw should, I suggest, be taken as affecting the monument at the third determination. On the other hand, if the record runs thus:

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[2] Investigation of Mining Subsidence at Barbauchlaw Mine, West Lothian. *Trans.* Inst. Min. Engrs. (1933) **85,** 303.

(1)	(2)	(3)	(4)	(5)	(6)	(7)	
0.00	0.01	0.03	0.01	0.03	0.04	0.08	etc.,

the line should be drawn at the fifth measurement.

There is agreement between the authors' observations, Auchmuty's[3] and our own respecting the initial phase of longitudinal movement: in each case it proceeded towards the mined area. In each instance, also, that swing was followed by another in the reverse direction. Auchmuty's results and ours at Barbauchlaw indicated that the return swing was bigger in magnitude and longer lived than the initial shift, with the result that the eventual displacement of the surface was, quite decidedly, towards the solid coal. I gather from Figs. 10 and 11 that the return movement was not so pronounced at Merrittstown and that, at the end of the tests, the monuments still remained displaced towards the gob. With this result in mind, may I ask an elementary question bearing on the horizontal movements of the shaft walls set forth in Table 2? The movements are stated to have been in the direction of the gob. From what standpoint was this? Was the shaft top or the shaft bottom taken as the fixed point?

The shaft measurements are especially interesting since nothing quite like them have previously been reported, so far as I am aware. I suggest, however, that an incorrect interpretation is placed in the results graphically depicted by Fig. 13. All through the paper, indeed, the authors refer to lines of break as though subsidence were solely a matter of fracture. Actually, much of the downward movement, and most of the effect of forward draw, is to be attributed to bending of the beds. The shaft observations showed that breaks were forming; but bending precedes fracture; and the figure is, I think, erroneous in showing lines of break as flattening out or as suggesting a "curved line concave to the coal bed." The results are perfectly normal in indicating that subsidence (from bending) occurs in the zone of draw well in advance of the fractures. In this connection may I call attention to Fig. 9*a*. Each of the three curves commences fairly smoothly and then passes into a vigorously oscillatory phase. It seems probable that the change is due to subsidence mainly caused by bending being followed by subsidence mainly caused by fracturing. If this view is correct, the breaks begin to occur 50 ft. in advance in a slow-moving face line and 90 ft. behind in a fast-moving face line.

While agreeing with the views expressed in the last paragraph of the paper, I nevertheless believe it to be more important to seek out the fundamental principles that apply to all cases of mining subsidence than to take the easier line of emphasizing the effects of local peculiarities, important though these certainly are. It is only by comparing such trustworthy records as may be available that those principles can be winnowed out.

H. J. HUMPHRYS,* Doncaster, Yorkshire, England (written discussion).—Subsidence due to mine workings in the rich South Yorkshire coal field is a matter of great importance since over large areas (200,000 acres) about three-fourths is below the 25-ft. contour. Some parts are only 6 to 8 ft. above Ordnance datum and a large portion of the area is below flood level in the rivers, and below the level of the water in the high-water carrier drains.

Mining operations are liable to derange the somewhat delicately balanced arrangements for surface drainage and sewage. The problem is complicated by the inadequacy of the river Don, and it was feared that large areas might be rendered derelict by the overflow of flood waters into basins created by mining subsidence, which could not be evacuated by gravity. In 1929 an Act of Parliament was passed setting up a special Authority to deal with the matter and to organize effective preventive and remedial works.

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[3] R. L. Auchmuty: Subsidence and Ground Movement in a Limestone Mine and on the Surface Caused by Longwall Mining in a Coal Bed Below. *Trans.* A.I.M.E. (1931) **94**, 27.

* H. M. Divisional Inspector of Mines.

In parts of the area 100 ft. of quicksand immediately underlies 8 ft. of subsoil and clay.

Careful records are kept by the various colliery companies, and through the kindness of Mr. C. W. Phillips, General Manager for Messrs. Barber, Walker & Co., Ltd., owners of Bentley colliery, I am enabled to give the following information:

"Records have been kept for 25 years of subsidence of the surface due to the working of the Barnsley Seam at a depth of 600–700 yd. within a few miles of Doncaster. The particulars given are in respect of longwall workings not less than 600 yd. wide and would not be true if applied to narrower excavations, such as a single conveyer face.

"The 'draw' in advance of a working face has been found to vary somewhat. With a face advancing at a normal rate of say 100 yd. a year draw occurs about 6½ chains in front of the workings and total settlement about 21 chains behind. If the rate of working is increased to a marked degree there is a 'lag' in the settlement. Draw over a rib side has been observed fairly consistently at a distance of 180 to 200 yards.

"Along the center of a panel of workings 600 yd. wide the maximum settlement of the surface was 3 ft. 9 in. with pronounced regularity but the thickness worked in that part of the mine did not exceed 4 ft. 9 in. Where the workings have been more extensive and the thickness extracted about 6 ft. (some top softs having been worked) the maximum subsidence is 4 ft. 10 in."

The right to work coal under valuable works in the Sheffield area was granted to a certain colliery by the Railway and Canal Commission on condition that the surface level should not be affected, and this condition has been substantially observed by leaving strips of coal unworked on a $^{53}/_{47}$ per cent basis, but the general practice is total extraction by a quickly advancing longwall face with good packing, valuable property being occasionally secured by iron straps and tie rods.

H. M. Hudspeth,* Sheffield, England (written discussion).—The behavior of the angle of draw is interesting. That the angle of draw increases when the rib line is stationary is what one might expect, but not so that a point on the surface within the zone of ground movement remains stationary during a face advance of 70 ft., except, as in this case, following a period during which the rib did not advance. A similar effect has been observed underground.

Again, the movement of the surface following upon the extraction of the coal is to a great extent a function of the nature of the intervening strata, and rocks that are able to sustain themselves with the aid of supports over large spans will result in differential subsidence values on the surface. In one case the overhang of the strata may be well within that which the rocks can sustain and the surface movements above this span will be small. When the span of strata is great enough to cause collapse there will be greater movements recorded on the surface.

Another factor to be considered is the effect of time on the breakdown of the rocks. It has been shown that rocks subjected to transverse loads will break down after a period of time under a load that was well within their strength at the instant of loading. Thus it is possible for a collapse of the strata to take place later than when a coal face is stopped, and this may explain the increasing angle of draw observed on a stationary face as shown in Fig. 4*b* Jan. 25 to Feb. 22. Collapse of the strata will then be followed by comparatively quiescent conditions on the surface until the span has again increased, so accounting for the decreasing angle of draw during the period March 8 to April 18.

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* Mining Engineer, Safety in Mines Research Board.

That the minimum rate of subsidence in advance of the face should be the result of a fast-moving rib line, and vice versa, is difficult to understand, and in this connection I would welcome information as to how the rate of advance was determined. The inclusion of the stoppage periods would explain this unexpected result. In the absence of differential subsidence, the rate of subsidence in advance of a face must increase with an increased speed of face advance over a long enough period.

The authors have rightly stated at the close of the paper that their results apply only to the particular conditions under which they were taken.

G. S. Rice,* Washington, D.C. (written discussion).—A number of problems in ground movement caused by mining are brought to attention in this paper:

1. Angle of draw produced by pillar extraction by a long although stepped break line, a method that is equivalent to longwall retreating without pack walls or back-filling.
2. The effect upon a concrete-lined shaft as a retreat line approaches, passes under and goes on beyond.
3. The effect on a large brick structure.
4. No "wave" of rise of the surface preceded the advancing subsidence.

This case again demonstrates that where there is a regular line of extraction there is a "positive" draw; that is to say, an effect of ground movement manifested on the surface above the solid coal (or essentially solid when less than 25 per cent has been mined) in *advance* of the extraction face. A positive draw is always found in longwall advancing, whereas in pillar withdrawal within panels surrounded by strong pillars, American experience has found the draw is often "negative."

The authors adopt the viewpoint, with which I fully agree, of considering the angle of draw as the angle between a vertical line above the point of extraction and an inclined line from this point to the farthest place on the surface in the direction of the advance, where a measurable subsidence or horizontal movement is found by precise survey and leveling. Very properly they also point out that the angle of draw, as thus defined, is not coincident with the irregular curve of shear, break, or bend of overlying beds between the same points.

In discussing the angle of draw, they have brought out that it is necessary to adopt for the surface point of advance subsidence some specific measurable minimum of movement, horizontal and vertical. They have adopted in the Merrittstown case 0.05 ft. for the vertical movement and 0.03 ft. for the horizontal movement.

Prof. Henry Briggs, in his discussion of this paper, suggests that there should be some standardized criteria in determining the angle of draw and suggests that the Institute committee consider adopting as its standard 0.03 ft., or about ⅓ in., for both vertical and horizontal movement. The standardization of some such minimum coming from so eminent an authority on subsidence as Professor Briggs is well worthy of the consideration of the Committee. Such a small movement as 0.03 ft. would hardly be appreciable in an ordinary structure, although it might affect machinery installations.

Professor Briggs questions the use of the term "line of break" employed by the authors. I agree that the term is not a satisfactory one although it has been frequently used, because, as he states, some strata bend or deform and may not fracture or shear. We will have to seek some better term. Tentatively I suggest the more lengthy term "line of initial bend or break." The succession of events as I visualize them from the bottom up are:

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1. The immediate roof layer breaks at the face as a cantilever.

2. Above the immediate roof the rock layers successively bend, or else, according to their nature, break initially as a cantilever, and the stratigraphic position of this initial bend or break will be affected by the thrust of arching stresses in the broken subsided overburden above the goaf. Whether any stratum will break, bend or deform will depend upon its rigidity. If a massive sandstone, for example, it will probably break as a cantilever or, in mining away from a boundary pillar, as a beam; a clay shale is likely to shear but a laminated sandy shale may bend where there is some degree of support.

3. As subsidence reaches the surface, if there is a thick bed of clay or soil this tends to slide, but initially the top layer may form vertical breaks to the surface. Therefore I visualize a very irregular line for break, bend or deformation, which is coincident with the line of draw only at the mine face and the surface.

Professor Briggs comments that no pre-elevation of the ground was observed in advance of the subsidence. It has been my experience that such pre-elevation occurs only where there is a very strong massive stratum near the surface such as sandstone or limestone with natural joints far apart forming blocks which successively balance on the edge of the solid strata below, the ends towards the advancing face tilt downward with the subsiding ground while the end over the solid tilts upward temporarily and lifts the earth above it. It is, I think, a rather rare phenomenon but something akin to it was indicated by the concrete foundation of the school house, the ground movement effects on which are described in the Newhall-Plein paper.

Colonel Rhodes says, in reference to the subsidence described in the paper by Horsley and Rhodes, that the seam is 49 in. thick, was mined by the longwall method, and the subsidence in the tunnel was 68 per cent of the thickness of the coal removed. Whereas at Merrittstown the subsidence from mining a seam 85 in. thick was only 55 per cent of the coal bed mined, at least up to the time when the last surveys were made. It seems probable that over a long period there may be some slight further subsidence at Merrittstown. In the case cited by Colonel Rhodes, the pack walls from roof brushing were constructed as is usual in the advancing longwall method. At the Merrittstown mine, only broken roof material and possibly 10 per cent of coal is left in the goaf. It is suggested that if sufficient geologic time elapses and under pressure of overlying rocks, surface subsidence, due to a removal from a given area of a volume of mineral or rock, will be practically 100 per cent of the thickness extracted. However, within the limited time since extensive mining has been done and records kept, it is found that percentage of subsidence in different coal fields, has varied widely due to these factors:

1. Extent to which pillars or blocks of coal have been left in place, intentionally or accidentally.

2. The degree of upsetting and breaking of the roof rocks, which varies with the method of mining. This, for example, is much greater in pillar withdrawal, especially when there are strong rock layers in the roof, than in advancing longwall mining where the roof is usually shaly and slowly subsides as the packwalls compress under load, generally without upsetting the broken roof blocks and fragments.

3. Strength of the broken rock and mineral left in the goaf in resisting deformation and compacting to their original density, by the closing of voids or spaces. For example, a clay shale will flow under very ordinary pressure as it does in a few years in the northern Illinois coal mines employing the longwall-advancing method under covers of only 300 to 500 feet.

4. Depth of cover or weight of the overburden. The greater the weight per unit of area over a large area of excavation, other things equal, the more rapid will be the subsidence.

In the cases under consideration, the High Hazel seam (Yorkshire, England) as referred to in Colonel Rhodes' discussion, averaged 4 ft. 1 in. thick and was mined by advancing longwall with pack walls. Apparently the immediate roof was largely shale, the thickness of cover below the tunnel was 613 to 701 ft. and with a maximum cover over the tunnel of 57 ft., making the average cover about 750 ft. The thickness of the Pittsburgh bed that was excavated and removed at the Merrittstown shaft, averaged about 7 ft. thick and was mined by a stepped-pillar retreat method without packing. Necessarily this method would upset pieces of broken roof and fragments of this may be considerably stronger than those of the immediate roof of the High Hazel seam. The thickness of the overburden at the Merrittstown shaft was 320 ft., which is less than one-half of the overburden of the High Hazel seam. These three factors, method of mining, strength of broken rocks in maintaining voids, and comparative weight of overburden may account for difference in subsidence percentage of the volume of coal removed in the two cases.

F. W. Newhall and L. N. Plein (written discussion).—Colonel Rhodes reports that the results given in the Merrittstown case largely confirm his experience; except that he noted a total subsidence of 68 per cent of the thickness of the coal removed, or 56 per cent of the thickness of the strata mined; whereas we report a subsidence of 55 per cent. Mr. Rice in his discussion has given an explanation of this difference in which we concur. We wish to emphasize especially the upsetting of broken roof fragments as roof falls occur following the extraction of pillars and pillar timbering in the Merrittstown case. Part of the immediate roof at the Republic mine consists of 2 to 3 ft. of a hard sandy shale, locally called a block shale because it breaks in large dimensional blocks. This particular stratum would be an important factor in maintaining voids and thus tending to reduce the percentage of subsidence. In the comparison of these two cases the percentage of extraction is a factor in the percentage of subsidence. While exact figures are not available we are safe in assuming that extraction of the High Hazel seam was probably 95 per cent complete whereas at Merrittstown it was somewhere between 70 and 75 per cent complete. We agree with Colonel Rhodes that the working of a seam 4 ft. thick, lying 265 ft. below the High Hazel, and mined 25 years previous to the mining of the High Hazel, could not have influenced any subsidence recorded in the mining of the High Hazel seam. We have had the opportunity to read the full paper by Messrs. Rhodes and Horsley, together with the interesting discussion that accompanied it, but time and space does not permit of any lengthy review of that paper; however, we believe that some interesting data would have resulted if levels had been taken on the surface overlying the tunnel at the same time that levels were taken in the tunnel. Information somewhat comparable to our shaft records would have resulted, especially with respect to the line of initial bend or break.

Professor Briggs asks what was considered as the fixed point for the shaft measurements. This question is answered on page 63, in the paragraph near the bottom of the page. It was realized at the beginning of the study that the shaft top could not be used as a fixed point because it would be subject to movement early in the study. Professor Briggs raises a problem of fundamental importance, which we believe is well worth discussing at length; that is, the question of the accuracy of surveys. He used a value of 0.02 ft. as indicating initial subsidence; we used 0.05 ft. Both are probably correct under the particular circumstances. It is our belief that a value should be used comparable with the errors of closure of the leveling surveys, that is if the errors of closure average 0.05 ft. it would not be proper to use 0.02 ft. as indicating initial subsidence. As indicated on page 64, our average error of closure was 0.011 ft., two of these were 0.032 and 0.030 and all the others 0.021 or less. We were perfectly safe therefore, in using 0.05 ft.; we might have used a finer value as 0.02 or 0.03 disregarding the three surveys in which the errors of closure exceeded 0.020 or the one that exceeded 0.030.

Two other factors should be considered in determining the accuracy of the work: (1) the time interval between surveys and (2) the distance between monuments. With weekly surveys on monuments 10 to 15 ft. apart, it would be best to use 0.02 ft. as initial subsidence; but if monuments 100 ft. apart are surveyed monthly even the value 0.05 ft. may be too precise; in fact, the results of work with such limits would be far from precise.

Each of our surveys required from 5 to 8 hr. of field work, depending on weather conditions, and we accepted the first run of levels for each survey as correct regardless of the closing error. Working in a time of depression with a greatly reduced engineering staff, we could not afford to spend more time in the field work. We feel that 0.02 is the lowest limit possible to use and 0.05 should be the upper limit. A compromise at 0.03 ft. should be equitable because an accuracy of that order is not difficult to obtain with reasonable cost.

Professor Briggs indicates his choice of readings in determining the point of initial subsidence. The following indicates our method, which is somewhat similar to that used by Professor Briggs. The values given in Table 3 do not represent any particular data resulting from our work but are hypothetical to indicate various possibilities that we found. The footnote indicates why results are shown giving the same angle of draw for 0.03 and 0.04 ft. subsidence. Our values of 0.05, 0.04 and 0.03 ft. do not mean subsidence equal to or greater than these values, but values close to these, picked with judgment with respect to results shown by near-by stations for preceding and succeeding survey dates. The average of initial subsidences used are 0.054, 0.039 and 0.029 ft. The point we wish to make is that in our work it was not always possible to pick the exact point of initial subsidence; therefore, personal judgment became necessary in making an arbitrary choice.

TABLE 3.—*Hypothetical Values**

Date	Station				
	0 + 00	0 + 25	0 + 50	0 + 75	1 + 00
	Total Subsidence, Ft.				
May 16	0.00	0.01	0.02	0.04[a]	(0.05)
June 1	0.02	0.05	0.03[b]	(0.04)[a]	0.06
June 13	0.02	0.03[b]	0.03[a]	(0.06)	0.08
June 27	0.03[b]	0.04[a]	(0.05)	0.08	0.10
July 11	0.04[a, b]	(0.04)	0.05	0.09	0.11
July 25	(0.05)	0.08	0.09	0.11	0.13

* The values in parentheses were used for initial subsidence of 0.05 ft. The values marked [a] are those used for 0.04 ft. subsidence; and the values marked [b] are those used for 0.03 ft. subsidence.

To determine what the angles of draw would be for 0.03 and 0.04 ft. we have computed Table 4. We do not feel safe in using 0.02 ft. because the error of our first survey to determine the initial position of our monuments was 0.020 ft.

As a basis of future studies we would set the following rules as necessary to obtain reliable results:

1. Surveys should be sufficiently accurate so that determinations may be made within the range of 0.02 to 0.05 ft., preferably 0.03 feet.

2. Surveys should be made at least once a month, but preferably twice a month or once a week if possible.

TABLE 4.—*Angles of Draw for Initial Subsidences of* 0.03, 0.04 *and* 0.05 *Feet*

Date, 1932	Line A				Line B			
	Rib Line at Station	Angles of Draw			Rib Line at Station	Angles of Draw		
		0.03 Ft.	0.04 Ft.	0.05 Ft.		0.03 Ft.	0.04 Ft.	0.05 Ft.
Feb. 8	3 + 02	22° 36′	19° 39′	18° 35′	3 + 20	31° 24′	24° 27′	21° 43′
Feb. 22	2 + 89	34° 22′	31° 16′	27° 55′	2 + 59	26° 21′	22° 45′	18° 50′
March 8	2 + 15	24° 06′	24° 06′	20° 17′	2 + 59	29° 44′	26° 21′	22° 45′
March 21	2 + 15	24° 06′	24° 06′	20° 17	2 + 30	25° 42′	22° 03′	22° 03′
April 4	2 + 15	31° 04′	27° 41′	24° 06′	2 + 30	32° 18′	29° 08′	25° 42′
April 18	2 + 15	31° 04′	27° 41′	24° 06′	2 + 30	25° 29′	29° 08′	25° 42′
May 2	1 + 50	[a]	25° 19	21° 37′	1 + 55	[a]	21° 51′	18° 00′
May 16	1 + 50		[a]	25° 19′	1 + 55		25° 29′	21° 51′
June 1	1 + 23			[a]	1 + 55		[a]	21° 51′
June 13					1 + 17			15° 50′
June 27					1 + 17			19° 47′
July 11					1 + 17			20° 59′
July 25					1 + 17			23° 41′
Aug. 8					1 + 17			23° 41′
Aug. 22					1 + 17			25° 01′
Sept. 6					0 + 47			[a]
	Line C				Line D			
Feb. 8	2 + 79	21° 18′	17° 33′	17° 33′	3 + 06	21° 23′	21° 23′	17° 45′
Feb. 22	2 + 79	24° 53′	21° 18′	21° 18′	2 + 41	15° 09′	15° 09′	11° 09′
March 8	2 + 70	30° 20′	27° 04′	23° 36′	2 + 20	19° 31′	15° 44′	11° 46′
March 21	2 + 57	28° 38′	25° 16′	21° 41′	2 + 20	19° 31′	15° 44′	11° 46′
April 4	2 + 40	26° 26′	22° 49′	19° 07′	2 + 20	19° 31′	15° 44′	11° 46′
April 18	2 + 00	20° 34′	16° 44′	12° 44′	1 + 90	14° 54′	10° 54′	6° 47′
May 2	1 + 60	21° 59′	18° 14′	14° 18′	1 + 75	16° 26′	16° 26′	12° 30′
May 16	1 + 60	21° 59′	18° 14′	18° 14′	1 + 75	20° 11′	16° 26′	12° 30′
June 1	1 + 60	21° 59′	21° 59′	18° 14′	1 + 75	20° 11′	16° 26′	16° 26′
June 13	1 + 60	21° 59′	21° 59′	18° 14′	1 + 75	23° 44′	16° 26′	16° 26′
June 27	1 + 37	22° 13′	18° 31′	18° 31′	1 + 75	27° 06′	23° 44′	20° 10′
July 11	1 + 26	20° 35′	16° 48′	16° 48′	1 + 75	27° 06′	23° 44′	20° 10′
July 25	1 + 26	[a]	[a]	20° 35′	1 + 75	[a]	[a]	23° 44′
Aug. 8	1 + 26			20° 35′	1 + 75			27° 06′
Aug. 22	1 + 26			20° 35′	1 + 75			27° 06′
Sept. 6	1 + 23			[a]	1 + 48			23° 24′
Sept. 19					0 + 91			[a]

SUMMARY OF ANGLES OF DRAW

Line	0.03 Ft.	0.04 Ft.	0.05 Ft.
A	34° 22′	31° 16′	27° 55′
B	32° 18′	29° 08′	25° 42′
C	30° 20′	27° 04′	23° 36′
D	(27° 06′)[b]	(23° 44′)[b]	27° 06′
Average angle	32° 20′	29° 09′	26° 05′
Maximum angle	34° 22′	31° 16′	27° 55′

[a] Subsidence beyond 0 + 00 station.

[b] Not to be considered as maximum angle of draw for 0.03 and 0.04 feet on D line, not used in average values.

3. Monuments should be set at distances not greater than 25 ft. apart on a line perpendicular to the line of retreating (or advancing) longwall or rib line.

With respect to horizontal movement, surveys made subsequent to the writing of the paper indicated that the monuments were all moving back towards their original location.

Our last paragraph was intended to be merely cautionary. If several other studies similar in character can be made under other conditions and the results check each other, then from all the studies general conclusions can be drawn regardless of specific conditions. Note that Mr. Hudspeth agrees with us in the statement that individual studies are affected by local conditions.

The fact that our angles result from a moving rib line should not be overlooked. We believe our maximum recorded angles are close to the maximum angle of draw that would result after final subsidence was complete but we cannot be positive of the degree of magnitude.

In reply to the discussion by Mr. George S. Rice, there is nothing much that we can add because Mr. Rice has been in intimate contact with the problem from the start and throughout he has furnished valuable guidance. To further supplement Mr. Rice's approval of our definition of angle of draw we quote from the reply by Colonel Rhodes and Mr. Horsley to the discussion of their paper: "The definition of draw adopted by us was that laid down by Dickenson, who defined it as 'The plane distance on the surface beyond the vertical projection of the line of face to the extreme point where the subsidence affects the surface.' This definition is supported by Lane and Roberts in 'The Principles of Subsidence and the Law of Support in Relation to Colliery Undertakings,' wherein draw is stated to be 'The distance on the surface by which the prime face is in advance of the vertical face.'"

The logic of Mr. Rice's theory of the succession of events of subsidence appears very reasonable to us in the light of our experience at Merrittstown.

It is important to emphasize Mr. Rice's statement with respect to the rise preceding the subsidence at the high-school building. As stated in our paper, the building rests directly on the Waynesburg sandstone, which is 60 ft. thick. At the shaft location this sandstone is entirely eroded and no rise in advance of subsidence was recorded at that point. It is our opinion that a massive stratum, with natural joints far apart, will materially affect the type of initial surface movements.

Mr. Hudspeth's query with reference to rate of subsidence and rate of movement of rib line is explained on pages 69 and 70. We have differentiated between rates of subsidence in advance of and in back of the moving rib line. In interpreting the rates of subsidence, the fact must not be overlooked that the rib line moved at varying speeds, and often did not move for many weeks at a time. Mr. Hudspeth's theory that strata may act as a cantilever and not break until the length of overhang becomes too great explains some of the results that we obtained.

Mr. Humphry's values at the Bentley colliery indicate positive angles of draw varying from 11°30′ to 13°30′ in advance of the moving face, and 16°30′ on the sides of the panels. In comparison with results at Merrittstown these angles appear to be too low and the question arises as to what decimal part of a foot was considered as initial subsidence and how often the surveys were made.

Prevalence of Anthraco-silicosis among Hard-coal Mining Employees*

By R. R. Sayers,† Associate Member A. I. M. E., and Roy R. Jones‡

(New York Meeting, February, 1935)

It has long been common knowledge that workers in anthracite are prone to develop a disabling disease of the lungs. Some of the earliest scientific contributors dealing with anthracosis were: Pearson[1], in his report in 1813, on the coloring matter of the bronchial glands and black spots in the lungs; Stratton[2], in 1838, who first termed the condition anthracosis; Greenhow[3], who reported to the pathological society in 1865, on the Black Lung of the Collier, and Pick[4] in 1870, who reported on the chemical composition of the ash of a collier's lungs. Visconti[5], in 1870, is accredited with being the first to use the term "silicosis" to denote the pathological condition of the lungs resulting from the inhalation of silica. Rovida[6] confirmed the reports of Pick and Visconti in 1871, in his report on a case of silicosis and chemical examination of the lungs for silica content.

The relationship of pulmonary infection, as associated with disabling pneumoconiosis affecting the coal miner, has frequently been discussed. While most reports have emphasized the role of infection, there has been some disagreement regarding the prevalence of tuberculosis among those suffering from anthracosis.

The report by Wainwright and Nichols[7] in 1905 has frequently been cited to show that the incidence of pulmonary tuberculosis among coal miners was less than that among the general population. It was their opinion that pulmonary tuberculosis was more prevalent among the miners' families than among the miners themselves. Collis and Gilchrist[8], in 1928 in their study of the Effects of Dust upon Coal Trimmers, concluded that among such workers there was an excess of deaths from bronchitis and pneumonia but not from pulmonary tuberculosis. They state, however, that "X-ray examinations of these workers reveals signs similar to those widely regarded as characteristic of silicotic fibrosis."

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* Based upon a review of the Report on Anthraco-silicosis (Miners' Asthma): A Preliminary Report of a Study made in the Anthracite Region of Pennsylvania, U.S. Public Health Service. Manuscript received at the office of the Institute Feb. 18, 1935.

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[1] References are at the end of the paper.

Later reports, such as that of Cummins[9] on Anthracosis, Silicosis and Tuberculosis; Badham[10] and his associates in their preliminary account of the chemical analysis and pathology of the lungs of coal miners in New South Wales; Sladden[11] in his report of the Silica Content of the Lungs, and Williams[12] in her report on the health of 100 old retired coal miners, all have caused continued consideration of the harmful effects of coal dust and the incidence of pulmonary tuberculosis among coal miners.

In 1924–25 the Division of Industrial Hygiene and Sanitation of the U.S. Public Health Service[13] made a study in two hard-coal mines in Pennsylvania, which indicated the existence of a hazard to health. Beginning in May, 1933, the Public Health Service[14] carried out a more extensive study dealing with miners' asthma as it affects the hard-coal mining employees. The need of such a study was emphasized by the report of the Special Commission[15] appointed by Governor Pinchot in 1932 to make a survey of occupational disease hazards in Pennsylvania. It was the opinion of the Commission that a study should be made regarding the exact nature and prevalence of chronic incapacitating miners' asthma affecting the coal miners of Pennsylvania, and it recommended that a survey be made by the U.S. Public Health Service in cooperation with the employers and employees in anthracite regions.

Plans for the Survey

Owing to the fact that the preliminary plans for actual conduct of the study played so important a part in its successful completion, a brief report of the drafting of the plans is submitted.

Following Governor Pinchot's request of the Surgeon General to make a survey the problem was discussed with the Pennsylvania State Department of Labor and State Department of Health. A conference was called in Washington on May 15, 1933, at which representatives of the state, Governor's Commission, operators and United Mine Workers were present.

The representatives of the State Departments assured the active cooperation of the various state labor and health agencies. They offered access to records and the aid of institutional staff members when and wherever necessary.

The operators granted access to employment records, entrance to the mines selected for study, and guides to accompany Service officers in their study of working conditions underground. Also, any of their company personnel and equipment necessary in the conduct of the study were to be made available.

The United Mine Workers' representatives pledged that all workers requested would appear for examination, and that committeemen would be detailed from each local to arrange a schedule for physical examination of the workers at the mines surveyed, and to act as interpreters when necessary for securing complete records.

The Service agreed to detail men for the work who were specially trained for the particular tasks and to make an analysis and report of observations.

It was further agreed that all records obtained were to become the property of the Public Health Service and that in no instance would the findings in any specific case be revealed to either the individual or his employer.

Three mines were selected for study as being representative of the mines and methods of mining common to the industry. All employees, office, surface and underground, were to be examined at each of the three mines surveyed. This arrangement avoided any criticism as to the type of mine or class of employee subjected to study, and at the same time furnished "control" mining methods and occupational groups for comparison with processes liberating excessive dust and workers employed under such conditions.

General Methods and Procedure

Actual field work started in May and continued through September, 1933. Supplementary studies such as that upon the health of disabled former anthracite workers were continued through the first few months of 1934.

Engineering.—The survey of each mine consisted first of a general mine inspection made for the purpose of obtaining information in regard to the mining methods used in the production and preparation of coal for the market; next, of the preparation of an occupational analysis to determine the number of employees and their classification according to occupational groups, at the mine being studied. Table 1 shows the number of employees by specific occupations in the three mines. It also shows the concentration of dust in the atmosphere to which the workers in the various occupational groups were exposed (weighted average).

Ventilation observations were made in the various work places. Although temperature and relative humidities were found to be fairly constant throughout all the workings, and were practically the same in each mine over the period studied (average dry-bulb temperature 58.5°F. and relative humidity 92 per cent), there was a considerable variation in the air movement in different work places. It was found that each mine was supplied with more than 200 cu. ft. a minute per man, as required by the Anthracite Mine Law. Although about 50 per cent of the workings investigated in this study had more than 50 ft. per minute air velocity at the face, many were practically dead ends, the air movement being

10 lin. ft. per minute or less. In such places it was found that the ventilation was improved by the use of compressed air.

TABLE 1.—*Summary of Occupational Dust Exposure of Workers*

Section and Occupation	Number of Men Exposed at Time of Survey	Number of Samples	Dust Count, Millions of Particles per Cubic Foot of Air (Weighted Average)[a]
INSIDE WORKERS			
Cutting and loading:			
Contract miners and laborers	1219	114	
Chamber mining	653		480
Breast (pitch)	204		203
Breast (scraper)	362		88
Breast (shaker)			55
Company miners	130		[b]
Chute loaders and starters	45	12	291
Shaker loaders	7	4	26
Scraper loaders	12	4	3.8
Scraper loaders engineer	31	[c]	3.2
Opening work (dry process):			
Rock drillers	95	17	241
Rock loaders (muckers and slatemen)	35	2	531
Transportation:			
Motormen, et al.:			
Mine No. 1	166	8	71
Mine No. 2	44	8	233[d]
Mine No. 3	69	13	3.1
Loader and driver bosses, stablemen, and helpers	10	[c]	6.9
Shaft, slope and plane workers, spraggers and couplers:	91		
Mine No. 1		7	25
Mines No. 2 and 3		2	3.1[d]
Other inside workers:			
Ventilation, timbering, and hoisting water section	276	[c]	6.9
Superintendence	36	[c]	2.9
OUTSIDE WORKERS			
Preparation (breaker):			
Dumpers and plane tenders			
Dry	2	2	71
Wet	7	7	14
Platform men and chippers			
Dry	7	2	69
Wet	23	7	24
Jig tenders and cone attendants	15	6	11
Slate pickers			
Dry	40	21	380
Wet	17	5	6.9
Car loaders et al.			
Dry	8	2	22
Wet	20	4	2.3
Other workers			
Dry	13	[c]	331
Wet	49	2	2.0
All other workers	386	34	2.9
Totals	2853	283	

[a] With impinger, U.S. Public Health Service method.
[b] Dust exposure depends on type of work performed as given in detailed occupational history.
[c] Dust count obtained from samples used for other occupations of similar exposure.
[d] Average includes other samples.

Dust Concentration.—For the purpose of determining the occupational dust exposure of mine workers, 300 atmospheric dust samples were obtained. A summary of the results of these determinations is presented in Table 1. It is evident from the data in this table that, with the exception of rock-working operations, chamber and pitch mining and chute loading constitute the dustiest occupations in the mining of anthracite coal. Chamber miners were found to be exposed to an average of 480 million particles per cubic foot of air, pitch miners to 203 million and chute loaders working in conjunction with pitch miners to 291 million particles. In connection with the chute-loading occupation it was found that motormen, because of their presence during the loading process, were exposed to an average dust count of 233 million particles per cubic foot. The rock loaders were found to be exposed to dust concentrations of 531 million particles, and rock drillers to 241 million particles. The other occupations in the mines involved exposure to varying dust concentrations, ranging from an average of 88 million for miners using scraper loaders to about 3 million for men in supervisory positions. From the data shown in Table 1 it is evident that 1076, or 47 per cent, of the 2266 inside workers in the three mines were exposed to more than 200 million particles of dust per cubic foot of air.

Table 2 shows the activities and associated dust exposures of chamber miners and laborers. Each activity engaged in by these coal miners was studied separately. Experience has shown that the various operations comprising the process of any dusty occupation are usually associated with dissimilar dust exposures.

TABLE 2.—*Dust Exposure of Contract Miners and Helpers (Laborers), Chamber Mining*

Activity	Number of Samples	Number of Hours in Activity	Average Dust Count, Millions of Particles per Cubic Foot of Air	Millions of Particle-hours per Cubic Foot
Jackhammer drilling	23	1	575	575
Hand loading	22	2	1138	2276
After firing	7	¼	834	209
Tamping and wiring	2	½	40	20
Setting up props, and in main airways	8	2¾	15	41
Totals	62	6½	....	3121

$$\frac{3121 \text{ millions particle-hours per cu. ft.}}{6\frac{1}{2} \text{ hr.}} = 480 \text{ millions particles per cu. ft.}$$

It is apparent from this analysis that the hand loading of coal contributes about 73 per cent of the total dust exposure of chamber miners and laborers, although this occupation comprises less than one-third of the total working hours. Jackhammer drilling, which occupies but 1 hr. of the 6½-hr. miners' day, amounts to 18 per cent of the total exposure, while the practice of entering a room too soon after blasting unduly exposes miners to a large amount of dust, this quantity being about

7 per cent of the total dust exposure. Drilling and loading coal, which involve slightly less than one-half of the miner's time in any one day, are responsible for 91 per cent of the total dust exposure.

The most dusty occupations in surface activities were found in connection with a dry breaker. As shown in Table 1, slate pickers and certain other workers were exposed to about 380 million particles per cubic foot. In contrast with such high dust concentrations, it was found that no group of workers in wet breakers was exposed to an average of more than 24 million particles per cubic foot.

Taking the dust data as a whole, 39 per cent of the workers were exposed to more than 200 million particles of dust per cubic foot; 62 per cent of the employees were exposed to more than 50 million particles per cubic foot, and 38 per cent to less than 26 million particles per cubic foot.

Individual's Total Dust Exposure.—In order to estimate the total dust exposure of each worker during his entire mining history, weight was given to the number of years and the average dust count associated with each of his mine occupations. It may be argued that the present dust counts are not representative of conditions that existed in the earlier years of a miner's occupational life. Certain observations on this point, however, indicate that the present dust determinations are fairly representative of conditions that have existed for a number of years in the anthracite mines. The jackhammer was first used in coal mines in 1909 and has been used very extensively since 1915. With the exception of the very recent improvements in mechanical loading and the use of wet methods in cleaning coal, conditions may be said to be about the same today as 20 or 30 years ago. One of the mines under study had an old-type dry breaker and handled coal both inside and outside the mine in a manner similar to that practiced many years ago. A comparison of the present

TABLE 3.—*Example of Method Used in Determining an Employee's Total Dust Exposure*

Employee's Occupation	Number of Years in Occupation	Average Dust Concentration, Millions of Particles per Cubic Foot of Air	Millions of Particle-years per Cubic Foot
Slate picker (dry breaker)	2	380	760
Patcher (dry mine)	2	71	142
Mule driver (dry mine)	3	71	213
Miners' laborer (chamber)	3	480	1440
Miner (chamber mining)	15	480	7200
Section foreman	5	7	35
Totals	30		9790

$$\frac{9790 \text{ (millions of particle-years per cu. ft.)}}{30 \text{ (number of years)}} = 326 \text{ millions of particles per cubic foot}$$

dust results with those obtained in a similar investigation made in this industry in 1926[13] show practically the same average results.

A typical example of the method used in arriving at a worker's dust exposure is presented in Table 3, in which the occupations are arranged in the order of employment, the last one being the worker's present occupation. It is obvious that if this last occupation only had been considered, the dust exposure for this individual would not have reflected the true state of affairs. In the above technic, weight is given to the number of years spent in each occupation and the amount of dust associated with each occupation.

Dust Composition.—Nineteen samples of dust that had settled out at the breathing level were collected from representative work places. These samples were analyzed both chemically and petrographically. A summary analysis giving the composition of the samples taken is shown in Table 4. In that table, miners and miners' laborers and the breaker workers make up the group exposed to dust containing less than 5 per cent quartz. Rock workers and all other inside workers, constituting the other two groups for which samples were taken, were exposed to dust containing about 35 and 13 per cent quartz, respectively. Because the amount of quartz found in dust has been generally used as a criterion of its relative harmfulness, the workers were classified into groups that depended on the percentage of quartz present in the dust as well as its concentration. Table 4 also shows that the free silica content of the samples increased with the inorganic matter present and the total combined silica. Alumina, while not given in the table, was found to a small extent (6 to 9 per cent on chemical analysis) in all the samples, except those pertaining to rock workers, in which it was quite high (23.5 per cent). In every instance the free silica found in the coal dust did not exceed 5 per cent.

TABLE 4.—*Composition of Samples Taken for Analysis*

Occupational Groups	Constituents Found by Computation from Results of Chemical Analysis, Per Cent			
	Coal	Other Mineral Matter	Total Silica in Rock and Coal	SiO_2 as Quartz (Av. of Chemical and Petrographic Analysis)
All miners and laborers	89.5	10.5	11.1	3.1
All breaker workers	84.2	15.8	13.4	4.4
All other inside workers	57.6	42.4	33.7	13.0
Rock workers	6.7	93.3	63.2	35.1

Particle Size.—The measurement of 1500 particles of dust, obtained by sampling the atmosphere in the breathing zone of various workers, revealed that the particles were practically all less than 4 microns in longest dimension, and that only 8 per cent were smaller than ½ micron. Eighty per cent of the dust was between 0.5 and 1.5 microns. The median size was 0.9 micron.

Dust Control.—During the course of this study it was possible to investigate coal-mining operations in which methods were being used that aided in the control of the dust hazard, as well as those which did not. A summary of the contrasting results obtained in the present study is presented in Table 5.

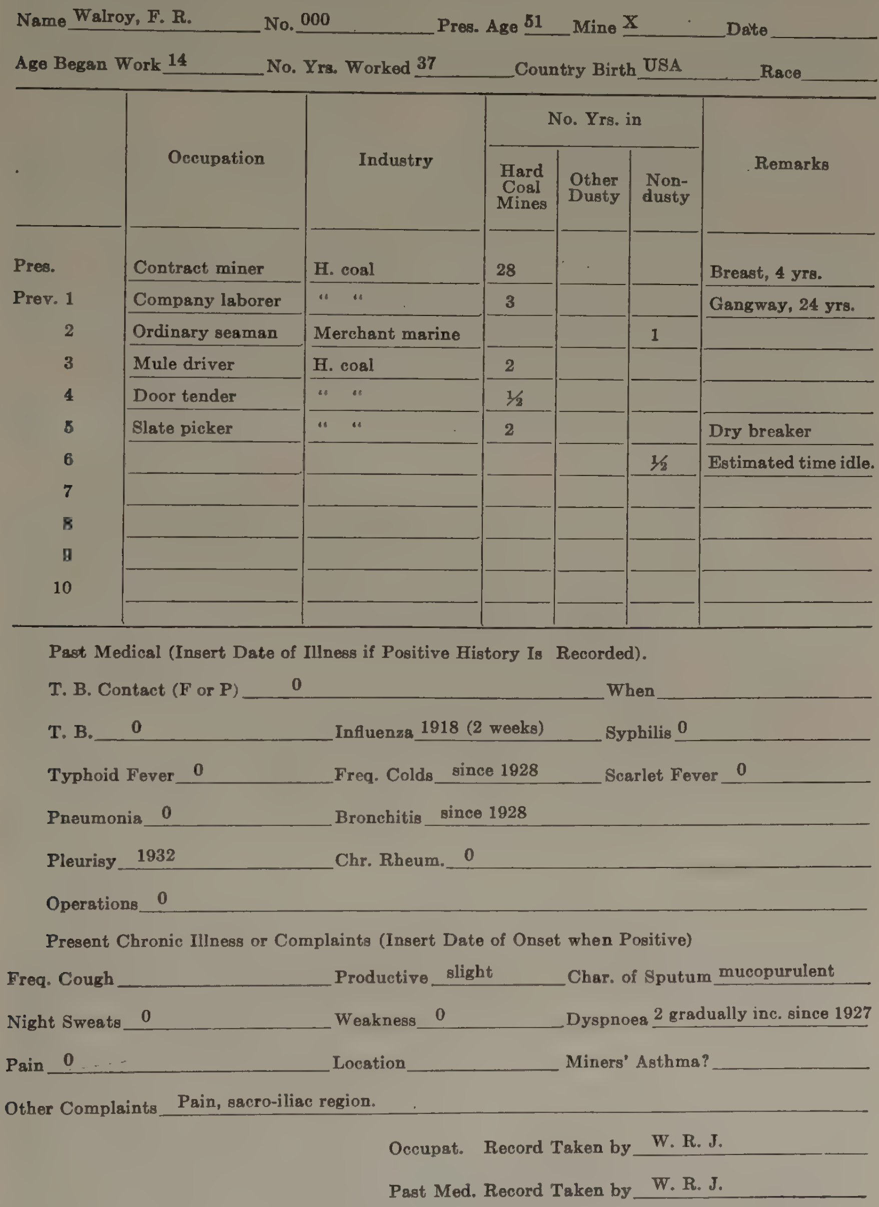
TABLE 5.—*Summary of Results Contrasting the Dust Exposure of Mine Workers under Controlled and Uncontrolled Working Conditions*

Operation	Dust Concentration, Millions of Particles per Cubic Foot of Air		Remarks
	Controlled	Uncontrolled	
Firing charge	40	834	Unless an adequate time interval elapses after firing a charge, miners found to be exposed to high dust concentrations.
Loading coal or rock	32	636	By wetting the loaded material the dust count is reduced as shown.
Loading coal	4–26	291–1138[a]	Mechanical loading decreases the dust exposure as indicated.
Drilling	33	568	Wet drilling is effective in reducing the dust concentration. Further reduction would necessitate exhaust ventilation.
Hauling coal in mines	1.2	17	Wetting coal and empty cars reduces dust in haulageways.
Preparation of coal	24	380	Wet breakers reduce dust counts as shown.

[a] The lower result is associated with hand loading of wet coal while the higher average is based on the loading of dry coal.

Medical Records.—Detailed occupational histories were recorded as shown in Fig. 1. This record of employment made it possible to classify workers into various exposure groups. The histories of past illnessess and their present complaints afforded a basis for a statement regarding the symptomatology of miners' asthma and were useful in completing the final diagnoses of the individual cases. Twenty-seven hundred and eleven men, or 96 per cent of the total number employed at the three mines, were examined by medical officers.

The physical examination was chiefly directed to measures designed to detect evidence of pulmonary changes. It consisted of a careful



Name Walroy, F. R. No. 000 Pres. Age 51 Mine X Date

Age Began Work 14 No. Yrs. Worked 37 Country Birth USA Race

	Occupation	Industry	No. Yrs. in			Remarks
			Hard Coal Mines	Other Dusty	Non-dusty	
Pres.	Contract miner	H. coal	28			Breast, 4 yrs.
Prev. 1	Company laborer	" "	3			Gangway, 24 yrs.
2	Ordinary seaman	Merchant marine			1	
3	Mule driver	H. coal	2			
4	Door tender	" "	½			
5	Slate picker	" "	2			Dry breaker
6					½	Estimated time idle.
7						
8						
9						
10						

Past Medical (Insert Date of Illness if Positive History Is Recorded).

T. B. Contact (F or P) 0 When

T. B. 0 Influenza 1918 (2 weeks) Syphilis 0

Typhoid Fever 0 Freq. Colds since 1928 Scarlet Fever 0

Pneumonia 0 Bronchitis since 1928

Pleurisy 1932 Chr. Rheum. 0

Operations 0

Present Chronic Illness or Complaints (Insert Date of Onset when Positive)

Freq. Cough Productive slight Char. of Sputum mucopurulent

Night Sweats 0 Weakness 0 Dyspnoea 2 gradually inc. since 1927

Pain 0 Location Miners' Asthma?

Other Complaints Pain, sacro-iliac region.

Occupat. Record Taken by W. R. J.

Past Med. Record Taken by W. R. J.

FIG. 1.—OCCUPATIONAL AND PAST MEDICAL RECORD.

physical examination of the chest and an exercise test to determine whether the individual was physically fit for arduous tasks. The interpretation of this test was based upon the respiratory and cardiac response immediately following the exercise and after a two-minute rest interval. An X-ray examination of the chest was made of each individual. The fluoroscopic examination was used to detect any chest abnormalities and film examination was made whenever changes were exhibited by fluoroscopy, even in borderline cases, and in addition films were secured on individuals considered normal, for control purposes.

A group of 135 totally disabled former anthracite workers were hospitalized for more complete observation. In addition to examinations similar to those given the active workers, temperature records and laboratory data were secured. The laboratory study consisted of complete blood counts, white cell differentials, blood sedimentation rates, hemoglobin estimations and Wassermann reactions. Gross and microscopic urine analyses were made. Sputum of each individual was examined by special staining methods, by culture and guinea-pig inoculation.

Pathological studies were carried out of a limited number of specimens of lungs forwarded through the courtesy of physicians practicing in the anthracite region. Chemical examination of the lungs for silica content was also made.

A study of silica excretion in the urine of a group of 120 active miners and former anthracite workers was carried out as a supplementary study.

At the time the workers were being examined information was obtained from them regarding their fathers, as to how long they had worked in the anthracite mines, as to whether they were alive or dead, and if dead, the cause of death. Where the father had died in Pennsylvania, an effort was made to check the death certificates on file in Harrisburg. From these data it was possible to arrive at certain conclusions concerning the mortality rates due to respiratory disease among anthracite workers.

Correlation of Dust Exposure and Evidence of Pulmonary Changes

By analysis of the data at hand it was possible to correlate total dust exposure and total silica exposure with the degree of pulmonary changes manifested. In like manner similar correlations were possible between dust exposure and decreased capacity for work, and between exposure and evidence of pulmonary infection.

The various occupations were grouped into four chief classes, largely in accordance with the silica content of the dust associated with the occupation. A control group was composed of all employees working in an atmosphere containing less than 5 million particles of dust per cubic foot of air. These were considered as controls, since no cases of anthraco-silicosis were found attributable to such exposure. The four chief occupational groups were as follows:

A. A group of 1435 men exposed to an atmosphere containing more than 5 million particles per cubic foot of air, less than 5 per cent of which was in the form of free silica.

B. A group of 426 men (a subdivision of group A) composed of miners and their helpers exposed to an atmosphere containing more than 300 million particles, less than 5 per cent of which was found in the form of free silica.

C. A group of 602 men employed in the haulageways exposed to an atmosphere containing more than 5 million particles, about 13 per cent of which was free silica.

D. A group of 151 men classified as full-time rock workers or miners that had spent more than two years in rock work, exposed to an atmosphere containing more than 5 million particles per cubic foot, about 35 per cent of which was in the form of free silica.

For a more detailed analysis these four groups were further divided according to the amount of dust exposure into four subgroups for each main group as follows:

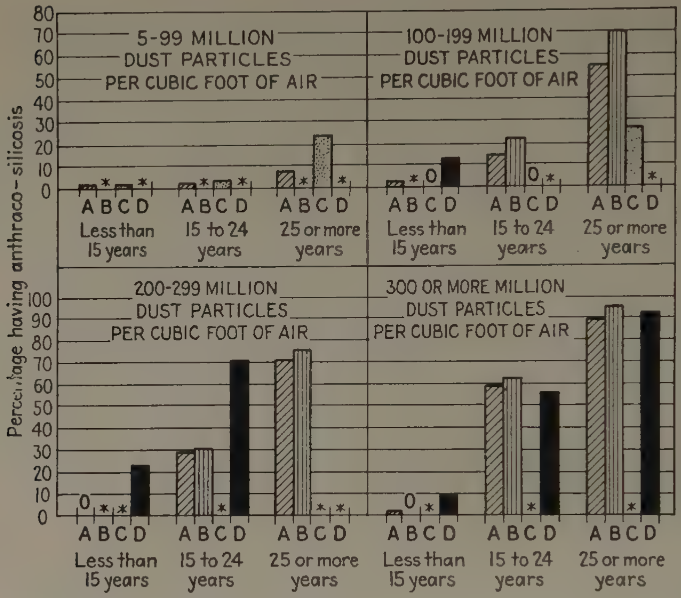
(1) Those exposed to from 5 to 99 million dust particles per cubic foot.
(2) Those exposed to from 99 to 199 million dust particles per cubic foot.
(3) Those exposed to from 199 to 299 million dust particles per cubic foot.
(4) Those exposed to 300 million and more dust particles per cubic foot.

Fig. 2 shows the percentage of men found to have anthraco-silicosis in the four chief occupational groups according to dust exposure and length of time in the industry. For each group there is a rise in the prevalence of anthraco-silicosis with the degree of exposure. Considering group A by subgroups, it will be noted that after 15 to 24 years exposure the incidence of anthraco-silicosis increases from 2 to 14 to 28 to 58 per cent. For this same group A, employed for more than 25 years, by subgroups, the increase in prevalence of anthraco-silicosis is from 7 to 71 per cent. A similar increase in the incidence with exposure is shown for the other groups. For group B this increase varied from about 20 per cent of those exposed to from 100 to 199 million particles for employment periods of between 15 and 24 years to more than 90 per cent when exposure was to over 300 million particles per cubic foot and the employment period 25 years or more.

The prevalence of anthraco-silicosis among rock workers (who had been exposed for more than three years to dust of which about 35 per cent was free silica) was greater than for other groups with corresponding lengths of employment.

Age *per se* appeared to play a minor role in the development of anthraco-silicosis. The part played by infection in relation to physical impairment is clearly shown in Fig. 3.

The marked excess in the prevalence of pulmonary infection in the exposed workers over those in the control is apparent for all periods of



**-Less than ten men examined in the group.*

O-No cases of anthraco-silicosis.

A-All emloyees except those in non-mining occupations underground and those who had worked in rock dust for more than 2 or 3 years.

B-Contract miners and helpers who had done little rock work. (Subdivision of group A)

C-Men in non-mining occupations underground except rock workers.

D-Men who had worked in rock dust for more than 2 or 3 years.

FIG. 2.—PERCENTAGE OF MEN HAVING ANTHRACO-SILICOSIS (STAGE 1, 2 OR 3) IN EACH OF FOUR GROUPS OF ANTHRACITE MINING EMPLOYEES, CLASSIFIED ACCORDING TO DUST EXPOSURE AND LENGTH OF TIME IN INDUSTRY.

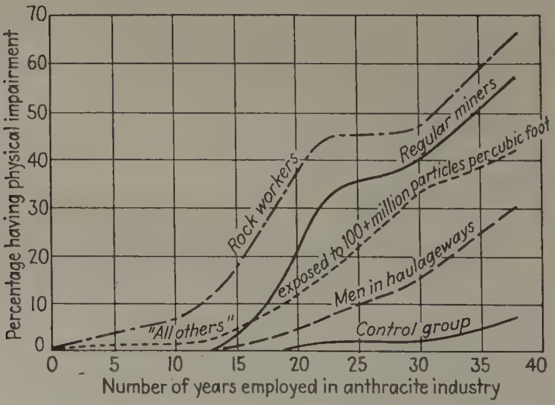


FIG. 3.—PERCENTAGE OF MEN HAVING PULMONARY INFECTION CONTRIBUTORY TO SLIGHT, MODERATE OR MARKED IMPAIRMENT CAUSING DECREASED CAPACITY FOR WORK, BY LENGTH OF SERVICE IN ANTHRACITE MINING INDUSTRY.

employment; but the rate of increase is much more rapid for those working in the more dusty atmospheres or where the silica content of the dust was highest. The highest rates of pulmonary infection were found among rock workers of more than 15 years service.

The percentage of workers having tuberculosis compared with the percentage of those among other industrial groups found to have the disease is graphically shown in Fig. 4.

The prevalence of pulmonary tuberculosis among the hard-coal mining employees at ages below 35 was not more than that in the other population groups referred to. In the age group 35 to 44, however, the prevalence of tuberculosis was about twice, at ages 45 to 54 about five times, and for the ages above 55 it was about ten times the rate found in the other groups. The fact that the silica content of the dust is largely responsible for this marked excess in the percentage of workers having tuberculosis is indicated in Fig. 5.

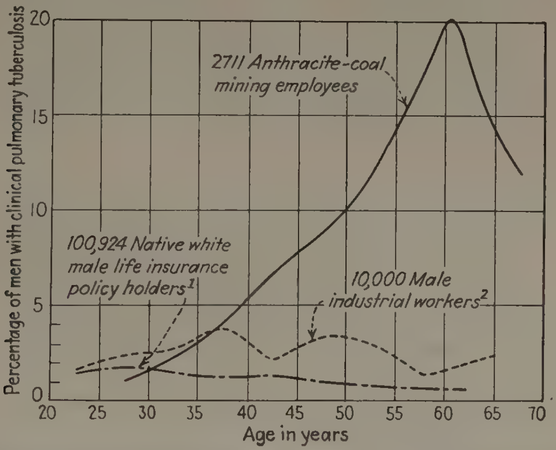


FIG. 4.—PERCENTAGE OF PERSONS HAVING CLINICAL PULMONARY TUBERCULOSIS, BY AGE, IN EACH OF THREE MALE GROUPS.

[1] From The Physical Impairments of Adult Life. *Amer. Jnl. Hyg.* (Jan. 1930) **11**, No. 1, 100.

[2] From A Health Study of 10,000 Male Industrial Workers. Public Health *Bull.* 162 (1926) 161.

The highest prevalence of clinical tuberculosis occurred among the rock workers. After 20 years' service, 37 per cent of these workers presented evidence of pulmonary tuberculosis.

When the workers were considered in groups, depending upon whether they were negative for anthraco-silicosis or were manifesting evidence of the disease in some stage, the role played by tuberculosis is further emphasized. Clinical tuberculosis was diagnosed in less than 1 per cent of the control group while for the entire 2711 workers such a diagnosis was made in 6.1 per cent of the cases. About 2 per cent of all those without anthraco-silicosis were diagnosed as clinical tuberculosis, about 15 per cent of these diagnosed anthraco-silicosis stage 1 were considered as evidencing clinical tuberculosis as a complication, and about 43 per cent of those with anthraco-silicosis in the more advanced stages presented evidence of tuberculosis as a complication.

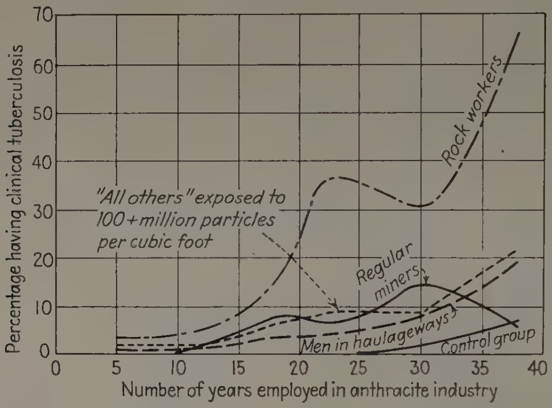


FIG. 5.—PERCENTAGE OF MEN HAVING CLINICAL PULMONARY TUBERCULOSIS, BY LENGTH OF SERVICE IN ANTHRACITE MINING INDUSTRY.

Fig. 6 shows the percentage of men evidencing physical impairment of any degree sufficient to decrease working capacity. The incidence of such physical impairments increase with the increase of dust in the atmosphere and with years of exposure. Likewise it was evident that those occupations where the silica hazard was greatest, considering total

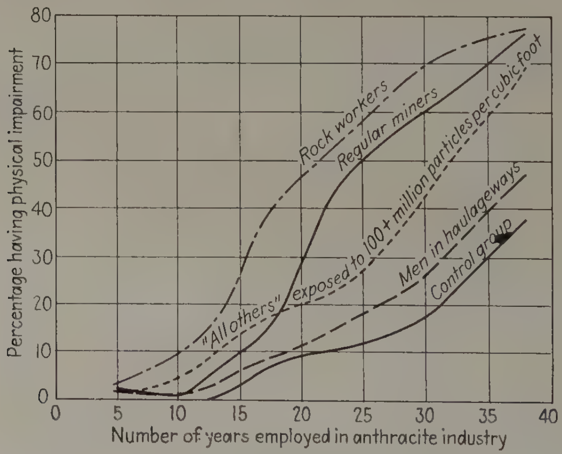


FIG. 6.—PERCENTAGE OF MEN HAVING SLIGHT, MODERATE OR MARKED PHYSICAL IMPAIRMENT OF ANY KIND SUFFICIENT TO DECREASE CAPACITY FOR WORK, BY LENGTH OF SERVICE IN THE ANTHRACITE MINING INDUSTRY.

concentration, were found to lead to greater percentages of physical mpairment.

In the control group 1.7 per cent were found with moderate or marked physical impairment causing decreased capacity for work as compared with 9.9 per cent among the regular miners, and with 12.6 per cent among the rock workers. With the exception of the rock workers, no group showed moderate or marked physical impairment in excess of that found among the controls when the period of employment was less than 20 years. However, it may be stated that an excess in the prevalence of slight impairment was found among the regular miners and among others in group A that had worked from 10 to 20 years in atmospheres containing more than 100 million dust particles per cubic foot.

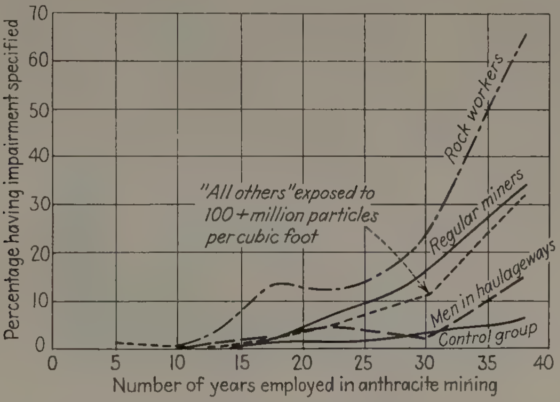


FIG. 7.—PERCENTAGE OF MEN HAVING MODERATE OR MARKED PHYSICAL IMPAIRMENT OF ANY KIND CAUSING DECREASED CAPACITY FOR WORK, BY LENGTH OF SERVICE IN THE ANTHRACITE MINING INDUSTRY.

Fig. 8 reveals the relative importance of different causes of disability at ages 55 and over, in specified groups of anthracite employees. The high rate of disability and the importance of pulmonary infection in contributing to it is obvious. The most striking feature, although one easily explained when the silica content of hard-coal dust is considered, is the role played by pulmonary tuberculosis in contributing to this disability in all the exposed groups.

The excessive mortality from respiratory diseases associated with anthracite mining is revealed in Fig. 9. It may be added that the rate increases with length of service and with silica exposure.

Such positive correlations between dust exposure and evidence of disabling miners' asthma leave little doubt as to the etiological significance of the dust in the air breathed. The term anthraco-silicosis is deemed appropriate as a descriptive title for miners' asthma as it affects the anthracite worker.

Anthraco-silicosis in the group studied was characterized anatomically, clinically, and roentgenographically by definite evidence of pulmonary changes. The complex nature of the disease makes it impossible to definitely diagnose the condition without evidence of dust exposure, positive clinical findings and definite pulmonary changes found upon X-ray examination. Border-line cases require repeated examination.

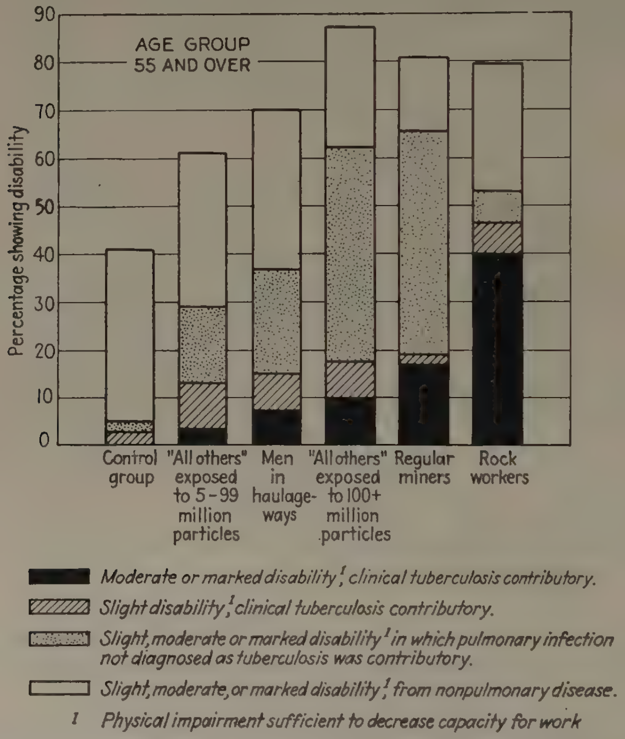


FIG. 8.—RELATIVE IMPORTANCE OF DIFFERENT CAUSES OF PHYSICAL IMPAIRMENT AT AGES 55 AND OVER IN SPECIFIED GROUPS OF ANTHRACITE-MINING EMPLOYEES.

Cases manifesting evidence of infection should be studied by laboratory methods. A post mortem diagnosis can be made by pathological and chemical examination of the lungs.

PREVENTION OF THE DISEASE

In regard to prevention of the disease it appears that exposure to atmospheres containing less than 50 million particles per cubic foot of air, where the free silica content is less than 5 per cent; to less than 15 million particles with about 13 per cent free silica; and to less than 10 million particles with a free silica content of about 35 per cent, would not produce disabling anthraco-silicosis or an excessive amount of chronic respiratory disease among anthracite workers.

It is evident from the engineering data that the levels of dust in the atmosphere indicated above have already been secured in some places by wet methods of mining and processing coal, adequate general and exhaust ventilation, and by the substitution of mechanical methods of loading. Because such a great amount of dust is liberated during blasting operations it is suggested that blasting be done only at the end of the shift.

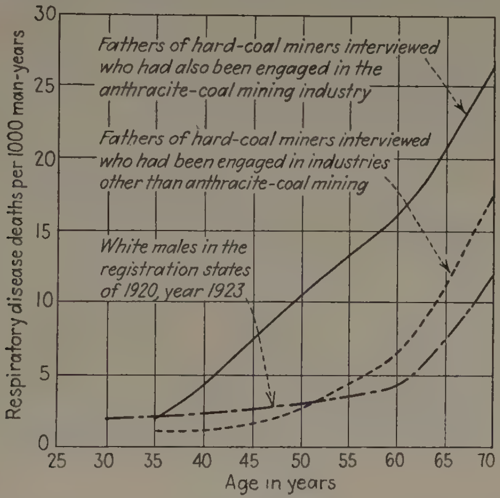


FIG. 9.—MORTALITY BY AGE FROM DISEASES OF THE RESPIRATORY SYSTEM, INCLUDING INFLUENZA AND TUBERCULOSIS OF THE LUNGS, AMONG THE FATHERS OF HARD-COAL MINERS INTERVIEWED.

Since infection has been found to have an important influence upon the course and development of the condition, none should be accepted for employment underground or in dusty occupations on the surface who are found to present evidence of pulmonary tuberculosis. For this reason all applicants for employment should be subjected to a careful physical examination that would include an X-ray examination of the chest. Furthermore, since many of those now at work may be more or less affected, all employees should be given an annual physical examination.

In the control of dust hazards there is no single measure applicable to all dusty operations and processes. All of the means of prevention mentioned must be practiced to insure success in the solution of the problem.

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Trend in Underground Lighting

By Graham Bright,* Member A.I.M.E.

(Pennsylvania State College Meeting, October, 1934)

Metal mines were developed long before coal mines and the early lighting of underground workings was effected by torches and candles. The early coal mines were outcrop workings and little trouble was experienced from gas, but as the mines became deeper and shaft mines were opened it became apparent that an open-flame lamp was exceedingly dangerous where gas occurred.

No satisfactory way of lighting gaseous coal mines was found until Sir Humphrey Davy brought out his famous safety lamp in 1815. The introduction of the Davy lamp was a great step forward but it was unsatisfactory from a light standpoint because the gauze absorbed a large percentage of the light given out by the flame. Clanny corrected this by the introduction of a cylindrical glass lens, which not only permitted a free distribution of the light from the flame but added greatly to the safety by eliminating the possibility of the flame coming in contact with the gauze and by protecting the flame against air currents.

A very complete story of the flame safety lamp since its inception is given in the U. S. Bureau of Mines *Bulletin* No. 227, by J. W. Paul, L. C. Ilsley and E. J. Gleim, published in 1924. A little more recent publication, known as Safety in Mines Research Board Paper No. 4, entitled "The Light Power of Flame Safety Lamps," by R. V. Wheeler and D. W. Woodhead, was published in England in 1927. The flame safety lamp was the only lamp available for gaseous mines for about 100 years.

Although the incandescent electric lamp has been in use over 50 years, it was not found safe to use this lamp from wired circuits in gaseous locations. The storage battery was advocated shortly after the development of the incandescent lamp but until the introduction of the tungsten filament in the miniature mine-lamp bulb, the battery lamp was not a real possibility because of the excessive weight of the battery necessary to produce a sufficient amount of light. The actual development of the electric-battery type of lamp took place largely between 1910 and 1915. In England and Europe, the development was almost entirely along the

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hand type of lamp whereas in the United States the cap lamp only was developed.

Of the many makes of cap lamps brought out in the early days, only one persists today. The credit for the development of a successful and practical electric cap lamp will go to Thomas A. Edison, who, largely from a humanitarian standpoint, spent a great deal of his time in the development of a safe and efficient lamp for the underground worker. Just how well he builded is witnessed by the fact that his lamp today is widely used in the mines of America.

Underground lighting in America at present is accomplished by two general methods. The lighting of shaft bottoms, main haulageways, switches, underground stables, machine shops, pump rooms, substations, etc. is usually accomplished by wired lights using power from the outside. This power may be taken from the direct current haulage and cutting system or from a separate alternating current system. The d.c. system is not well suited to this work, as the voltage regulation that is satisfactory from a haulage or cutting standpoint will as a rule be very unsatisfactory from a lighting standpoint. A 15 per cent variation in voltage is considered very satisfactory from haulage or cutting standpoint but for a lighting system it will mean a variation in light of 60 per cent. Since these variations in voltage take place rapidly, the resulting illumination is very unsatisfactory. On the other hand, an a.c. circuit can be installed in which the regulation can easily be kept below 5 per cent.

The voltage of the direct current systems in America makes it difficult to install an efficient lighting system. The voltage may be anywhere from 220 to 300 or from 500 to 600. With alternating current, 115 volts is usually selected, as this voltage entails the lowest first cost and permits the use of the most efficient bulb.

As it is almost impossible to obtain gas accumulations in dangerous quantities in the locations mentioned above, even in gaseous mines, it is deemed safe to install wired electric lamps with open wires or with wires in conduit.

Permissible wired lighting systems have been developed and are used to some extent abroad but are not used in this country. For illumination at the face of a coal mine where there is always danger of accumulations of explosive gases, the permissible electric cap lamp is used exclusively in America. The lighting at the face in metal mines and in so-called nongaseous mines until recently has been accomplished with open-flame cap lamps and hand lamps. The recent developments in connection with electric cap lamps in America have changed the situation to such an extent that these are coming into use in increasing numbers.

Since the wired lighting in use today in America presents no unusual features and involves no new developments over a period of years, this paper will be confined entirely to the portable type of lamps used by the miner at the face and others underground requiring a portable lamp.

General Lighting Terms

To discuss lighting conditions underground, it is well to first become familiar with some general lighting terms and lighting units.

Candlepower.—The standard measure of light is the candlepower, which is the light given out in a horizontal direction from a candle of certain standard dimensions.

Foot-candle.—The actual intensity of illumination on a surface of an object from a burning candle depends upon the distance the surface is from the candle. Let us assume that we have a source of light of small dimensions, giving one candlepower and located in the center of a sphere of which the radius is one foot. Every point on the used surface of the sphere will be just one foot away from the light source and the actual illumination on the surface of the sphere will be *one foot-candle.* If we substitute a sphere whose radius is 2 ft., the area of the inside of the sphere will be four times as great as before. The same amount of light will therefore be spread over four times the area. Consequently the intensity will be one-fourth as great. The intensity of illumination will therefore vary inversely as the square of the distance from the source of light. The actual illumination on any object or surface is determined not by the candlepower of the source of light but by the foot-candles of light falling upon the object or surface.

Lumen.—The candlepower and foot-candles are measures of intensity and give no indication of the actual volume or quantity of light given out by the source of illumination. If a surface of one square foot area is uniformly lighted with an intensity of one foot-candle, the total volume of light received by the surface is termed one *lumen,* which is the unit of light volume or light flux.

If a sphere of one foot radius has a source of one candlepower at the center giving off a uniform light in all directions, each point on the inside of the sphere will have an intensity of illumination of one foot-candle and each square foot of area on the inside of the sphere will receive a light flux of one lumen. The total area of the inside of the sphere is $4\pi r^2$ or 12.56 sq. ft. The total lumens given off by a source of one mean spherical candlepower will therefore be 12.56 lumens.

Mean Spherical Candlepower.—In order to determine the total light flux or lumens from any source of light, it is therefore necessary to determine the average candlepower in all directions. This average is termed the *mean spherical candlepower,* and usually is measured by a large sphere photometer. It is also susceptible to calculation from a light-distribution curve, as will be shown later.

Mean Candlepower.—It is sometimes desirable to know the average candlepower of a lighting unit where the light is given out at a definite zone, as with the electric cap lamp. The average intensity over the light stream of the zone is known as the mean candlepower. The mean candlepower, if the zone is 360°, is of course the same as the mean spherical candlepower. If the zone is, say, 180°, the total light will be concentrated in an angle of 180° instead of 360° and the mean candlepower will be twice as great as the mean spherical with proper allowances made for losses due to absorption or reflector efficiency.

Efficiency.—The efficiency of an electric light bulb is usually expressed in watts per mean spherical candlepower. A 2.4-volt, 1.25-amp. lamp bulb will have a watt consumption of 3 (volts multiplied by amperes). The efficiency of such a bulb is about 2 watts per spherical candlepower with a life of 250 hr. The mean spherical candlepower will therefore be 3 divided by 2, or 1.5.

The efficiency of an entire lamp is the percentage of the mean spherical candlepower of the bulb that is actually given out by the complete lamp. The efficiency of a complete electric cap lamp varies from 55 to 80 per cent.

The efficiency of the average British type of hand lamp is about 50 per cent. This low efficiency is due to loss in thick lens used, loss due to posts, and the fact that reflectors cannot be applied to collect the light in upper and lower zones.

Photometers.—Mean spherical candlepower is best measured by a spherical photometer and, for accurate results, the sphere should not be less than 30 in. in diameter, preferably larger. Light-distribution curves are best made in a laboratory using a high-grade photometer. The photronic cell, however, has simplified the determination of candlepower readings and added to their accuracy because of its simplicity, ease of handling, and elimination of the personal equation, and this cell can be readily made up in the form of a portable photometer for use in the field.

Present Recommended Standard of Illumination in Industry

Until recently the question of the proper illumination for industrial workers had not been given much thought. It was felt that too much light would be injurious to the eyes and it would be best to err on the low side. The human eye, however, is a wonderful organ in that it can adapt itself to utilize light over a very wide range of intensity. Sunlight will produce an intensity of 6000 to 10,000 foot-candles. A modern industrial lighting scheme will use 20 to 30 foot-candles, and although this illumination looks almost excessive when installed, the day is considered a dull one if the light from the open sky is less than

100 foot-candles. It is rather difficult to furnish too much artificial light provided it is directed properly and there is absence of glare.

The best recommended practice today for lighting in industry varies from 3 foot-candles, for the illumination in storage spaces and stock rooms for storage of bulk material, to 50, and even 100, foot-candles for fine assembly work, sewing of dark goods, engraving, etc. The minimum intensity of light recommended today is 2 foot-candles and, where such light is permissible, no close work is to be done. Eye strain will result when the illumination provided for ordinary observation is materially lower than 2 foot-candles. Owing to the wonderful recuperating power of the human eye, no serious results seem to occur with intensities as low as 0.1 foot-candles, but continuous work using illuminating values as low or lower than 0.1 foot-candle is almost sure to cause serious eye strain.

Types of Lamps Used in Europe and America

Mining in Europe is very much older than in the United States, and there is naturally a much stronger tendency toward conservatism and towards following a precedent in Europe than in America.

In general, the English and European miner prefers the hand lamp while in America the cap lamp is used exclusively in coal mines and to a large extent in metal mines. Some metal mines use large carbide hand lamps but they are gradually giving way to the electric cap lamps. The reasons given by the American miner for preferring the cap lamp are as follows:

1. Allows the free use of both hands at all times.
2. Directs the light just where the miner is looking.
3. A miner's light never shines in his own eyes and the direct light from other lamps in the vicinity does not seem to be objectionable with the matte surface reflector.
4. The actual illumination available, even if the hand lamp and cap lamp are placed at the same distance from the work, is about 14 times as great in favor of the cap lamp, and when, as is usual, the cap lamp is only one-half as far away as the hand lamp from the work, the difference is 56 times in favor of the cap lamp. The proof of this will be shown later.

The objections against the cap lamp, such as weight to be carried on the hat and belt, are inconsequential, as experience has proved that the miner soon becomes so accustomed to the evenly distributed weights that he forgets that he has them with him.

Some of the deep English coal mines are hot and the miner objects to wearing a hat or cap. The best safety practice at present indicates that all miners should wear protective hats or caps, and since these can be provided with ample ventilation there is no reason why a miner in the hot mines could not obtain the advantages of a cap lamp.

ILLUMINATION AVAILABLE TO THE UNDERGROUND WORKER IN THE PAST AND AT PRESENT

Great Britain

From the Thirteenth Annual Report of the Secretary of Mines for Great Britain for the year ending Dec. 31, 1933, the safety lamps in use by the underground workers in the coal mines can be classified as follows:

270,417	flame safety type
362,764	electric hand lamps
28,508	electric cap lamps
661,689	total

Cap lamps represent only 4.3 per cent of the total.

Most of the 28,508 electric cap lamps are found in Scotland, where the mine operators seem to realize the many advantages that can be obtained with them as compared with a hand lamp.

The flame safety lamps are the same that have been used for many, many years from a lighting standpoint and actual tests indicate that the large majority of these lamps will not average more than one-half a horizontal candlepower. New high-candlepower flame safety lamps have been developed recently in England that will give a mean horizontal candlepower of 3 to 4. Few of these, however, have been placed in service, so that it is not possible to state how successful they will be.

The single-cell battery hand lamps do not average over one candlepower whereas the two-cell lamp averages 1½ to 3 candlepower (horizontal candlepower).

When a hand lamp is used by a miner, he usually hangs it up on a post or sets it on the ground, and the distance of the light from the work is usually about 6 ft. Since the intensity of light varies inversely as the square of the distance from the source, the actual illumination on the work for such lamps will be as follows:

FOOT-CANDLES

$$\text{Flame safety lamp } \frac{0.5}{36} = 0.0138$$

$$\text{Single-cell hand lamp } \frac{1}{36} = 0.0276$$

$$\text{Two-cell hand lamp } \begin{cases} \frac{1.5}{36} = 0.0417 \\ \frac{3}{36} = 0.0834 \end{cases}$$

The minimum amount of light recommended for industrial applications when no real work is to be done is 2 foot-candles. The English miner using a flame safety lamp receives only about 1⁄150 of the minimum requirement. With a single-cell hand lamp, he received about 1⁄70 and with the two-cell lamps only about 1⁄30 of the minimum. When it is remembered that the exposed surfaces at the face of a coal mine are black and absorb possibly 80 to 85 per cent of the light that is received, the absence of real illumination is much more pronounced.

From this information, it is little wonder that the dreaded eye disease known as nystagmus is so prevalent in England. This disease, caused without doubt by insufficient light, costs the English coal operator over half a million pounds sterling each year in compensation. This disease is unknown in America and the question of compensation has never come up.

The British Mines Department has been aware for some time of the inadequacy of the illumination furnished underground and has been working for the past two years on a new set of Lighting Regulations, which should improve conditions to a considerable extent. These new regulations have been completed and very recently have been put into effect. Two years are allowed the coal-mining companies to replace lamps that do not meet the new regulations, which must, however, be met by all new installations.

The new regulations will mean the replacement of nearly all of the flame safety lamps in use at present and practically all of the single-cell hand lamps. Many of these, however, can be made to meet the new conditions by substituting a two-cell battery for the present single cell.

The following outlines briefly the United States Bureau of Mines and the British Mines Department Regulations:

U.S. Bureau of Mines Regulations

The essential requirements contained in the U.S. Bureau of Mines Schedule 6-B for the approval of the electric cap lamp for use in gaseous atmospheres underground are as follows:

1. Total angle of light flux from headpiece to be not less than 130°.
2. Average minimum candlepower delivered at any point in the illuminated area (taken at design voltage of bulb) shall not be less than one candlepower.
3. The maximum candlepower at any point in the light stream must not be greater than five times the average over the light stream. (This is to prevent the use of a concentrated beam giving a bull's-eye effect, and requires the use of a matte surface reflector.)
4. There shall be no sharp contrast over the light stream.
5. The battery shall have sufficient capacity to permit the bulb to operate for a 12-hr. period.
6. A 10-hr. bulb can be used to meet shorter working hours (a recent revision).
7. The bulb shall have a life of 200 hr. at the design voltage and 95 per cent of the bulb shall have a life of 150 hr. (To meet these conditions requires a design life of 250 hr.)

8. When the battery case is opened for charging, a magnetic lock is required.

9. A lock is required on the headpiece, and this is usually of a lead seal type.

10. All bulbs must be equipped with a safety mounting and when the bulb presses against the lens, the lens must be of safety glass.

British Lighting Regulations (Both Hand and Cap Lamps)

1. Total angle of light flux from headpiece of a cap lamp to be not less than 100°, and for a hand lamp 180°.

2. Mean spherical candlepower at end of 9 hr.: 0.75 for hand lamps, 0.40 for cap lamps.

3. Mean horizontal candlepower of hand lamp to be not less than 1.5 at end of 9 hours.

4. Bulb must have a design life of 600 hours.

5. Safety glass is required for a cap-lamp lens if the bulb is not equipped with a safety mounting.

The minimum spread of light of a cap lamp is 130° in America, and 100° in England. The concentration of the light flux in an angle of 100° will allow a much higher average intensity, but the advantages, if any, of this smaller angle can be determined only by a thorough trial.

The new British regulations permit the use of a reflector in connection with a hand lamp, the minimum angle of light being 180°. The design of a reflector for a hand lamp presents some very difficult problems since it is desirable that the illumination over the entire 180° be fairly uniform. It is comparatively easy to build up the illuminating values toward the center of the light stream with the natural forms of reflector, but to build up the outer zones in order to obtain fairly uniform light over the entire 180° is a real problem.

In America

In the Bulletin entitled "Coal," issued by the U.S. Bureau of Mines, the total number of underground workers in the coal mines of the U.S.A. in 1933 was 432,569. The total number of electric cap lamps in use in 1933 was 266,166.

The light available for the miner in America from the electric cap lamp from 1915 to the present time varies over a wide range, but even with the earliest lamps giving a beam candlepower of about 2, the illumination provided on the work was much greater than with the present improved hand lamps in Europe.

The headpiece of an electric cap lamp is not often more than 3 ft. away from the place where the miner is actually working. He has frequent occasion to look at objects more than 3 ft. away but not for the purpose of actual work.

Owing to the shorter working hours, a 10-hr. bulb is now available which gives about 20 per cent more light than a 12-hr. bulb.

The actual illumination in foot-candles available to the American miner while working at the face during the past 20 years is as follows (headpiece 3 ft. from work):

1914–1922	0.24	1927–1930	4.0
1922–1924	0.7	1930–1934	6.0
1924–1927	1.8	1934	7.32

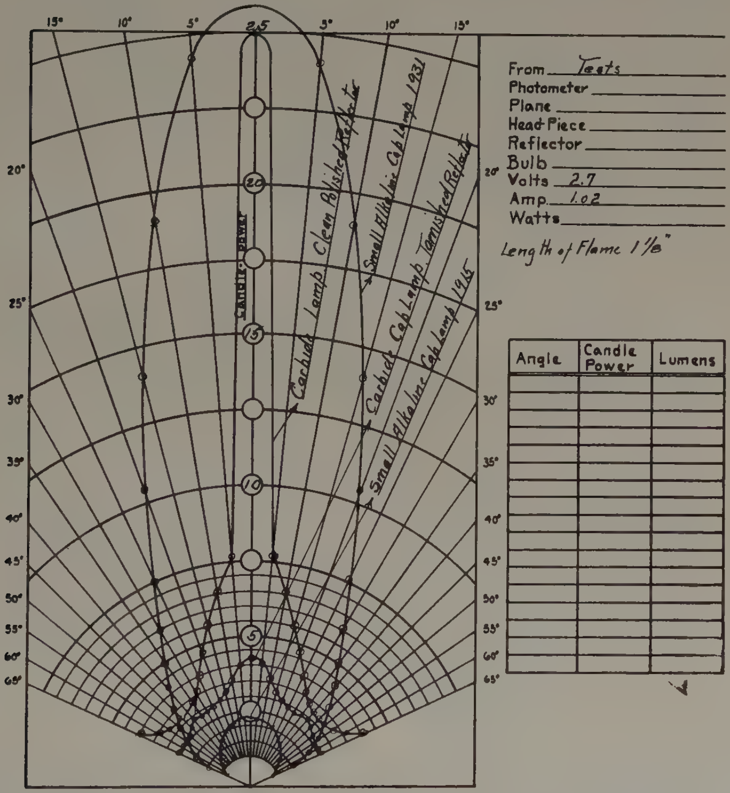


FIG. 1.—LIGHT-DISTRIBUTION CURVES OF SMALL CARBIDE CAP LAMP IN USE IN COAL MINES WITH CLEAN REFLECTOR AND WITH TARNISHED REFLECTOR, AND TWO TYPES OF TWO-CELL ALKALINE ELECTRIC CAP LAMPS.

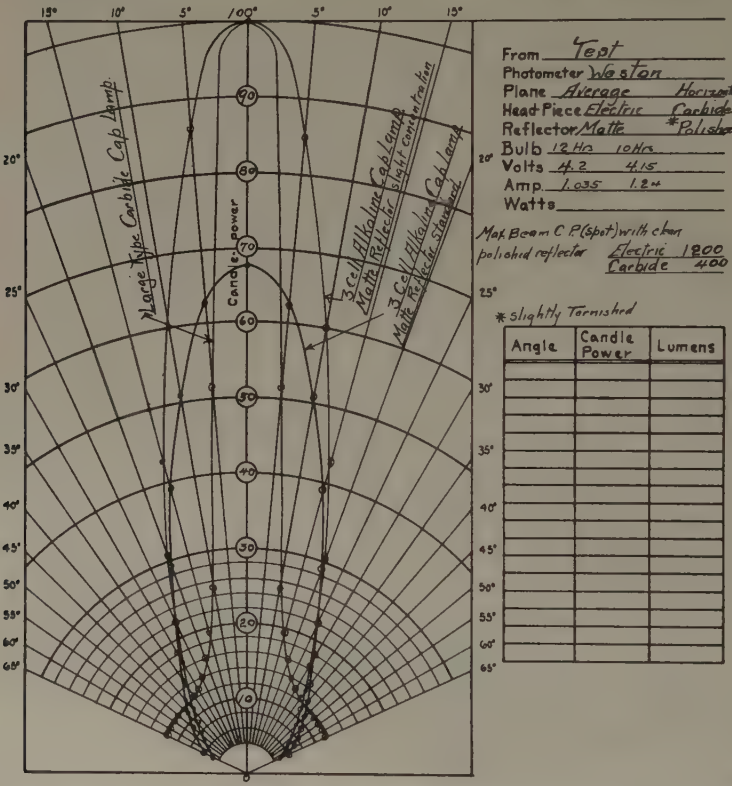
Even under the worst conditions the small lamp used 15 to 20 years ago gave more light than the English miner gets today with the best hand lamp.

The light available in America from the latest type of cap lamp is well above the minimum now being recommended for industrial work.

TYPES OF CARBIDE LAMPS

When the carbide lamp was introduced for underground lighting about 1910–12, it gave so much better light than the old oil torches and the first electric cap lamps that, unless a mine was considered gaseous, the carbide lamp was the only one considered. The carbide lamp is made in both the cap and hand type, the cap type being used exclusively in connection with coal mines and some metal mines, and the hand type in other metal mines.

All carbide lamps are equipped with reflectors but these are seldom kept clean, so that, with the exception of some of the large hand lamps, the light given out is all direct from the flame with no help from the reflector.



From Test
Photometer Weston
Plane Average Horizontal
Head Piece Electric Carbide
Reflector Matte *Polished
Bulb 12 Hrs 10 Hrs
Volts 4.2 4.15
Amp. 1.035 1.24
Watts

Max. Beam C. P. (spot) with clean polished reflector Electric 1200 Carbide 400

* slightly Tarnished

Angle	Candle Power	Lumens

FIG. 2.—LIGHT-DISTRIBUTION CURVES OF LARGE CARBIDE CAP LAMP AND THREE-CELL ALKALINE ELECTRIC CAP LAMP.

Light-distribution curves of the carbide lamp are shown in Figs. 1 and 2. These curves are taken in quiet air with a definite length of flame and with clean and tarnished reflector.

Curve 1 shows the light distribution from the small type of carbide cap lamp used in open-light coal mines. Tests were made with clean polished reflector and with the usual tarnished reflector. A spot light of 25 candlepower is obtained with the polished reflector but this dwindles to a little over 4 candlepower when the reflector becomes tarnished.

On the same curve is plotted the distribution curve of the early alkaline electric cap lamp showing the reason why this lamp could not compete with the carbide lamp from a standpoint of illumination. A fourth curve, of the smallest modern two-cell alkaline electric cap lamp, is also plotted in curve 1, and clearly indicates the superiority of the present electric cap lamp from an illuminating standpoint.

A larger type of carbide cap lamp is used in some of the metal mines. With a clean polished reflector, it will give a spot light of 400 candlepower. However, after a short time the reflector becomes tarnished and the beam candlepower will drop to 100 or less.

Curve 2 shows the light distribution of the large carbide cap lamp after it has been burning a short time, and this reflector has become slightly tarnished. On the figure will be found two light-distribution curves of a modern three-cell alkaline electric cap lamp using a 10-hr. bulb. One of these curves shows the distribution when using the standard reflector, in which the maximum beam candlepower is a little less than five times the average over an angle of 130°. The other curve shows the distribution when using a matte surface reflector with a little greater concentration in which the maximum beam candlepower is nearly seven times the average. The metal miners seem to favor the reflector having the greater concentration and it may be found that a ratio of greater than 5 to 1 will be more satisfactory for the worker in metal mines.

In curve 2, although both the carbide and the three-cell electric cap lamp with the more concentrated reflector have the same maximum-beam candlepower, the distribution at other angles up to 75° indicates the great superiority of the electric over the carbide. It will be noted that the increase from 68 to 100 candlepower with the electric lamp is obtained at the expense of a small loss in candlepower from 50° to 130°. This drop could scarcely be detected by the eye.

Further increases of beam candlepower can be obtained by using various finishes of the matte surface so as to give 200, 400, 600 candlepower until a polished surface is reached when a maximum beam candlepower of 1200 is obtained. The various values just mentioned can be maintained over a long period since the headpieces are dustproof and water-tight, so that the reflector surface will remain clean.

Types of Storage-battery Lamps

Storage-battery lamp applications are the most severe that the storage battery has to meet with in industry. The average miner is a husky individual who is used to working hard with simple strong tools that will stand rough treatment. He is likely to forget that an electric cap lamp will not stand indefinitely the rough treatment he sometimes gives it. It is not uncommon for the battery of a cap lamp to be used as a hammer to drive nails or to drive a wedge on top of a post. It is also frequently used as a pad between the miner's body and a mine-car frame when it becomes necessary to push a car.

Two types of storage batteries have been used from the beginning of the development of the electric cap lamp in America, the lead acid and the nickel-iron alkaline types.

Electric Cap-lamp Bulb

The bulb for an electric cap lamp is just as carefully designed and constructed as the larger bulbs used for our home and industrial lighting. In fact, it is more difficult to produce uniformity in regard to candlepower life, and efficiency due to the low voltages and short lengths of filaments involved. The bulb must be designed to meet the characteristics of the battery that supplies the power. The weight of such a battery will vary from 3½ to 5 lb., and the power available from such a battery for a 12-hr. period is from 25 to 45 watt-hours.

Light-life.—The actual illumination given out by an electric bulb depends upon the temperature at which the tungsten filament is operated. For many years all incandescent lamps were operated with the filament in a high vacuum. The actual temperature of the tungsten filament was found to be about 2500° F. for a reasonable life. With a vacuum bulb, an evaporation of the tungsten takes place which not only blackens the inside of the glass but also thins down the filament, which increases the resistance and consequently reduces the current and wattage. The result is a drop from 20 to 40 per cent of the amount of light given out by the bulb during its life.

It has been discovered that for certain types of bulbs, if the bulb is filled with an inert gas like argon the temperature can be raised 200° to 300° for the same life and with very little evaporation of the tungsten. The result is that there is practically no blackening of the bulb and a loss in light of only about 3 per cent during the life. At the same time there is a marked increase in light due to the higher temperature of the filament. The gas-filled construction presents no advantage unless the current rating is 0.6 amp. or above.

Efficiency.—The efficiency of an electric bulb is measured by the number of watts required to produce one spherical candlepower and depends upon the temperature of the filament. With voltage of 2 to 2.5, the filament is very short and only a small part of the filament midway between the mounting terminals can be operated at the correct temperature. The mounting supports draw considerable heat from the part of the filament that is adjacent to the supports and consequently reduces the temperature a marked amount. This, of course, reduces the light output so that the efficiency is adversely affected.

With a 3.5 to 4-volt bulb, the filament is longer and a much greater percentage of it is far enough away from the supporting posts to be unaffected. The result is that a very marked increase in efficiency can be obtained with the higher voltage bulb.

The efficiency of a bulb will increase a small amount with an increase in current rating, because the heavier filament required by the increase in current rating is sturdier and can be operated at a little higher temperature and still have ample mechanical strength. The heat losses, due to convection, will also be less due to the fact that the area of the filament per unit of volume is less than with a smaller filament.

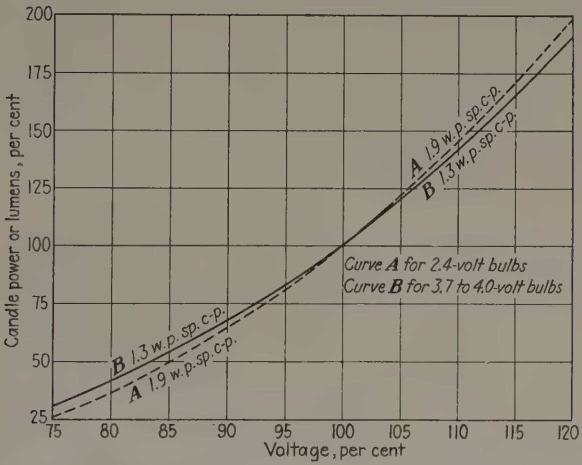


FIG. 3.—CHARACTERISTIC CURVES OF ELECTRIC CAP-LAMP BULBS (MAZDA C, GAS-FILLED) SHOWING VARIATION IN CANDLEPOWER AND LUMENS WITH CHANGE IN VOLTAGE.

The three-cell alkaline lamp, therefore, has a decided advantage over the two-cell lamp from an efficiency standpoint. (Figs. 4 and 5.)

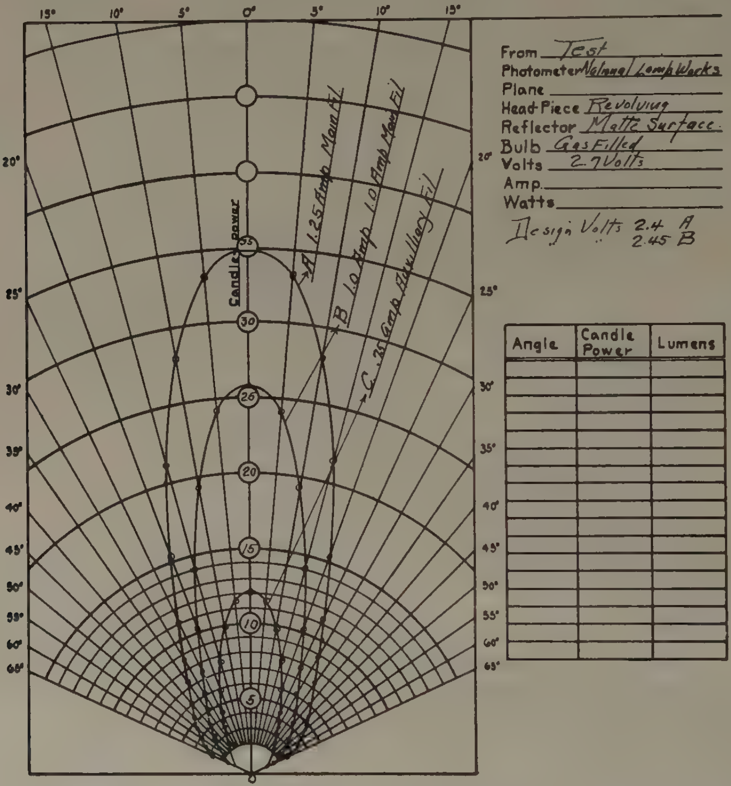
The early cap lamps were equipped with a single-filament bulb with no switch. A little later a lamp was brought out using two bulbs side by side, one bulb acting as a reserve. This lamp did not give a balanced light distribution. The next type used a main single-filament bulb and a smaller auxiliary bulb placed at one side. The disadvantages of this type were low headpiece efficiency, poor light distribution of auxiliary bulb, and high bulb maintenance.

In 1927, the two-filament bulb was developed for the electric cap lamp. The main filament was designed to use the full capacity of the battery while the auxiliary or minor filament was designed to take about 60 per cent of the current required by the main filament. The advantages of this type of bulb were:

1. Assurance of a working light for each entire shift.
2. No extra space required by auxiliary filament and consequently no sacrifice of reflector surface.

3. Minor filament being practically in the center of the reflector, it gave a balanced light distribution and could be well used as a working light. In fact, so well did this scheme work out that it was not at all uncommon for a miner to reach up and turn on his auxiliary filament and work all day with this filament without knowing it was the minor filament.

4. The minor filament, being very close to the main filament, is kept hot and will stand more vibration than when cold, consequently it is a simple matter to design the minor filament to last longer than the main filament.



From Test
Photometer National Lamp Works
Plane
Head Piece Revolving
Reflector Matte Surface
Bulb Gas Filled
Volts 2.7 Volts
Amp.
Watts
Design Volts 2.4 A
" " 2.45 B

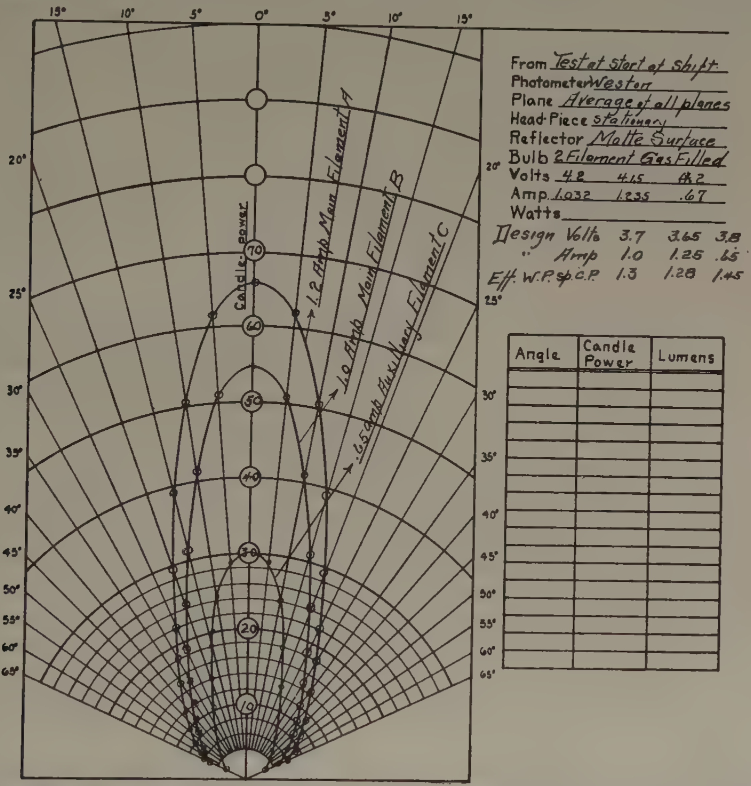
Angle	Candle Power	Lumens

FIG. 4.—LIGHT-DISTRIBUTION CURVE OF TYPICAL TWO-CELL ALKALINE ELECTRIC CAP LAMPS.
1.25 amp. 12 hr.; 1.0 amp. 12 hr. and 0.75 auxiliary filament.

5. The two-filament bulb will cost only a little more than one with a single filament and the maintenance will be much less than when using two separate bulbs. The main filament of a two-filament bulb is only slightly less efficient than a single-filament bulb.

An experiment is being tried using a two-filament bulb with each filament a main filament designed for one-half life. The bulb is used until both filaments are burned out, thus giving the normal life with an increase in efficiency of 3 to 5 per cent. An auxiliary bulb is used for emergency but cannot be used as a working light. It will be interesting to see how this rather complicated arrangement works out in practice.

Bulb Life.—A mine-lamp bulb is designed to have a life of 250 hours. Sometimes a somewhat greater life is obtained in the field. No doubt this is due to the fact that the average care of the lamps in the field is not as good as that received during tests in the laboratory. The bulb could be designed for a longer life but it would be at the expense of light, as the longer life would be obtained by lowering the efficiency. A shorter life would, it is true, provide more light but only at a greater cost and with a sacrifice of service. There seems to be little objection to renewing a bulb every three to four months but if it has to be done oftener the service suffers. A longer life would save in cost of bulb renewals but the saving would not be commensurate with the loss in light caused by the lowered efficiency.



From Test at start of shift
Photometer Weston
Plane Average of all planes
Head-Piece stationary
Reflector Matte Surface
Bulb 2 Filament Gas Filled
Volts 4.2 4.15 4.2
Amp. 1.032 1.235 .67
Watts
Design Volts 3.7 3.65 3.8
" Amp 1.0 1.25 .65
Eff. W.P. sp. C.P. 1.3 1.28 1.45

FIG. 5.—LIGHT-DISTRIBUTION CURVE OF TYPICAL THREE-CELL ALKALINE ELECTRIC CAP LAMPS.
12-hr. main, 10-hr. main and auxiliary filament.

In England the bulb is designed for a life of 600 hr. Although it is felt by many in England that the field life is the same as the laboratory life, the writer has come in contact with many operators in England and Australia who were very proud of the fact that it was not uncommon to obtain a life of a year or more from a hand-lamp bulb. This is a clear indication that there is a large sacrifice in light for a very small saving in maintenance cost.

Variation in Light with Change in Voltage.—A bulb is designed to give a definite spherical candlepower at a voltage known as the design voltage. A variation in voltage applied to the bulb will cause a greater variation in candlepower. Fig. 3 shows the actual change in candlepower for changes in voltage for both the 2.4-volt and the 3.7-volt bulb designed for 250-hr. life. If the efficiency of a bulb is known at normal voltage, the spherical candlepower, the beam candlepower, or the lumen values can be readily calculated for any other voltage.

Headpiece.—Much thought has been put on the design and construction of the headpiece of an electric cap lamp. The early headpieces were of metal or hard rubber. They were neither waterproof nor dustproof. Starting about 1927, the metal headpiece was replaced by one of molded Bakelite, which is strong, light in weight and nonflammable. It will readily hold inserts if required and has been made both water-tight and dust-tight. A modern headpiece weighs only 6 oz., so that it is not an uncomfortable weight to carry on the head.

The reflector is an exceedingly important part of the headpiece. The Bureau of Mines Schedule requires the reflector to have a matte surface to give a spread light. Matte surface reflectors are usually made of aluminum and, up to 1928, the matte surface was produced by sand blasting or by a caustic dip. At that time it was felt that the reflectivity or the efficiency of the reflecting surface was not satisfactory. An actual test made in the laboratories of the Aluminum Company of America showed that the reflector surface efficiency was only about 60 per cent and the efficiency of headpieces varied from 55 to 65 per cent, depending upon the amount of reflector surface available. Careful research work developed a new chemical finish that raised the reflector efficiency to 85 per cent. At the same time, the U.S. Bureau of Mines changed the schedule to permit the maximum beam candlepower to be five times the mean. Immediately these two changes made available a greatly increased amount of light in the actual working angle of the headpiece, which is about 15° on either side of the center or a total angle of 30°. The actual foot-candle available was more than doubled. The net result was an increase in the headpiece efficiency and a much better light distribution.

After a number of actual efficiency tests had been made on complete headpieces by the use of large spherical photometers, it was found that the efficiencies were susceptible to calculation if the efficiency of the bulb, the active area of the reflector, and the reflector efficiency were known.

The direct light from a headpiece is 28.8 per cent of the total light given out by the bulb. Of this direct light, 95 per cent of it is actually sent out, 5 per cent being absorbed by the lens. The other 71.2 per cent must be gathered up and reflected if it is to be used.

The older headpieces of the single-bulb type had a reflector efficiency of 60 per cent and of the 71.2 per cent of indirect light about 93 per cent was covered by reflector surface. With the two-bulb type, in which one bulb is large and the other small, the small bulb cut off a fairly large percentage of reflector surface, so that of the 71.2 per cent of indirect light only 70 per cent is covered by active reflector surface.

The following shows how closely the headpiece efficiency can be calculated.

1. Two-bulb headpiece (one large and one small auxiliary bulb):

Per Cent
28.8 per cent direct light, 5 per cent loss through lens = 27.35
71.2 per cent indirect light.
70 per cent effective area.
60 per cent reflector surface efficiency.
5 per cent absorption through lens.
Total effective reflected light =
71.2 × 0.70 × 0.60 × 0.95 = 28.4

Efficiency or total effective light = 55.75

A test of this headpiece in a spherical photometer indicated the efficiency to be 55 per cent.

2. Single two-filament bulb with high efficiency reflector:

Per Cent
28.8 per cent direct light, 5 per cent loss through lens = 27.35
71.2 per cent indirect light.
93 per cent effective area.
85 per cent reflector surface efficiency.
5 per cent absorption through lens.
Total effective reflected light =
71.2 × 93 × 85 × 95 = 53.5

Efficiency or total effective light = 80.5

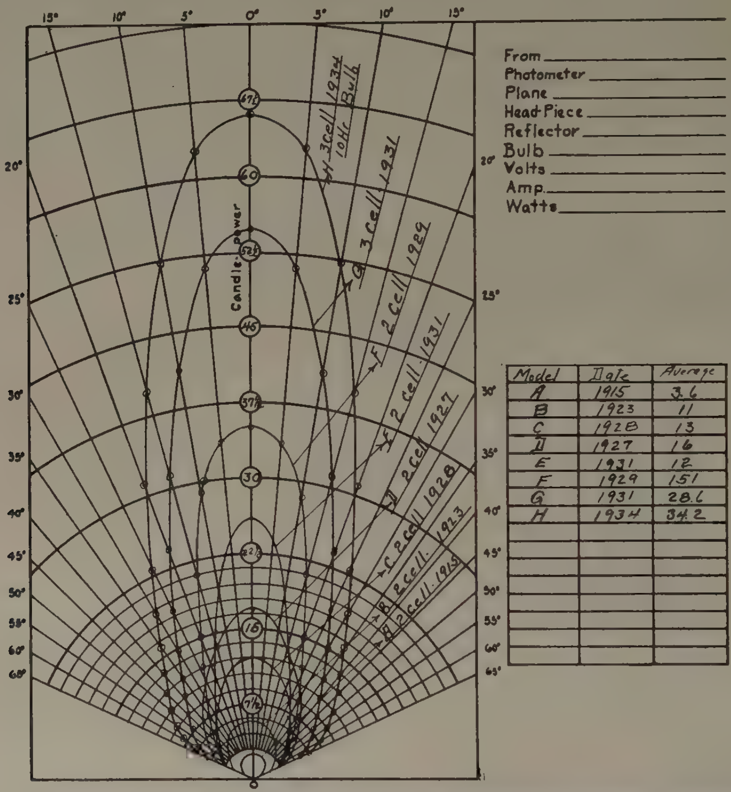
The actual efficiency as measured by a large spherical photometer averaged 80 per cent.

The angle at which the headpiece is mounted on the cap is very important. When the wearer is walking upright, the extreme lower edge of the light stream should be about at the feet. This will enable the wearer to walk along an entry with comfort and without fear of stumbling over obstacles, which he could not well observe if there were an area of darkness at the ground in front of him.

Light Distribution and Effective Illumination

The light-distribution curve of an electric cap lamp will tell more or less the complete story of the illuminating value. These curves are usually based upon the voltage at the beginning of the shift, at the middle, or at the end of a 9-hr. or 12-hr. shift. Light-distribution curves are shown in Figs. 1 to 5. A combined distribution curve in Fig. 6 shows progress in light distribution since 1915.

The three-cell alkaline lamp curves (Fig. 5) show the main filament of the 12-hr. and 10-hr. bulbs, and also the auxiliary filament. The distribution of the auxiliary filament produces almost as good a working light as the main filament of the two-cell lamp.



From
Photometer
Plane
Head-Piece
Reflector
Bulb
Volts
Amp.
Watts

Model	Date	Average
A	1915	3.6
B	1923	11
C	1928	13
D	1927	16
E	1931	12
F	1929	151
G	1931	28.6
H	1934	34.2

FIG. 6.—COMBINED CURVE, SHOWING PROGRESS IN DEVELOPMENT OF ELECTRIC CAP LAMP SINCE 1915.

Calculation of Lumens

Although it is difficult to measure spherical candlepower or obtain lumen values directly in the field, it is possible to calculate these values if the distribution curve is available. This curve can be taken in the field with a photometer. The tests should be made in several planes and the results averaged. Readings are usually taken every 5° from 0° to 65°, on either side of the center. If the lumen output from the headpiece is desired, calculations must be made to determine the total output of light in the entire angle of distribution of 130°.

Let it be assumed that the filament of the bulb is in the center of a hollow sphere whose radius is one foot. The illumination from the headpiece will then be represented by a cone of light whose angle is 130° and this light will illuminate a segment of the inside surface of the sphere whose area will be 3.6 sq. ft. (Since the area of the entire sphere is 12.57 sq. ft., the average intensity of the light over 3.6 sq. ft. will be 12.57 ÷ 3.6 or 3.5 times the mean spherical candlepower.)

Since the light given off by the headpiece is not uniform (nor is this desirable) it is necessary to determine the average illumination over the surface of the segment in order to determine the lumens or quantity of light.

The light given off by the center cone represented by a total angle of 10° at the center will illuminate a small circle on the sphere whose area is 0.0239 sq. ft. The average intensity of light on the circle would be represented by the candlepower at 2.5°. The lumens falling upon this circle would therefore be the candlepower reading multiplied by the area.

The light given off by the next 5° would be a hollow cone and would illuminate a zone on the inner surface of the sphere whose inner angle would be 5°, and whose outer angle would be 10°. The average intensity of light on this zone would be the candlepower at 7.5°. The area of this zone is 0.0715 square foot.

It will be easily understood that the area of each succeeding zone from 0° to 5° will increase a large amount from the center. The actual area of each 5° zone is shown in Table 1. The area of the zone from 60° to 65° is over 20 times as great as the center zone of 10°. It is readily seen that candlepower values near 65° represent very much more in lumens than the values near 0°. For this reason, the candlepower values in the actual working angle of approximately 15° on either side of the center can be built up a considerable amount to assist in making a real efficient headpiece from a practical seeing standpoint without taking an appreciable amount from the outer angles near 65°.

To obtain the total lumen output of a headpiece from a light-distribution curve, it is therefore necessary to determine the candlepower at the center of each zone 2.5°–7.5°–12.5°–17.5°, etc. Each candlepower reading is multiplied by the area of its zone, which gives the lumens in that particular zone. The sum of the lumens in all zones gives the total lumens. Since there are 12.57 lumens in each mean spherical candlepower, the actual mean spherical candlepower of the headpiece can be readily obtained by dividing the total lumens by 12.57.

Knowing what the spherical candlepower of the bare bulb should be, we can readily determine the efficiency of the complete headpiece, since these calculations will give the mean spherical candlepower of the headpiece.

TABLE 1.—*Light Distribution and Lumen Output of Electric Cap Lamp*

Readings taken at rated voltage of bulb.
Test bulb 1 amp., 3.7 volts, 3.7 watts.
Efficiency of bulb 1.31 watts per spherical candlepower.
Spherical candlepower of bulb, 2.825.

Angle	Candlepower	Zone Area	Lumens
2.5	34.2	0.0239	0.82
7.5	29.1	0.0715	2.08
12.5	22.8	0.1186	2.70
17.5	17.1	0.1648	2.82
22.5	13.	0.2098	2.72
27.5	10.1	0.2531	2.56
32.5	8.3	0.2940	2.46
37.5	7.02	0.3340	2.35
42.5	6.22	0.3700	2.31
47.5	5.2	0.4040	2.10
52.5	4.62	0.4350	2.01
57.5	4.12	0.4620	1.90
62.5	3.48	0.4860	1.69
		Total Lumens	28.52

Lumens from bulb = 2.825 × 12.57 = 35.45 at 3.7 volts.
Efficiency of headpiece = 28.52 ÷ 35.45 = 80.4 per cent.

Table 1 shows the complete set of calculations necessary to determine the lumen output and efficiency of a headpiece. The test-bulb information is given and its lumen output is 35.45 at the design voltage.

The calculated efficiency of the headpiece is 80.4 per cent, whereas the actual measured efficiency was 80 per cent.

Determination of Maximum Permissible Beam Candlepower

The actual shape and area of a light-distribution curve gives no indication of the total lumens given out by the headpiece. It was pointed out above that the actual lumen value in the central angle was very small compared with the same angle at 130°. Two distribution curves may give the same lumens whereas one may have twice the beam candlepower of the other one. The maximum beam candlepower that can be obtained from a headpiece and meet the U. S. Bureau of Mines Schedule is susceptible to an accurate calculation as follows:

The area of the part of a sphere represented by an angle of 130° is 28.8 per cent of the total area of the sphere. This means that the area of the sphere would be 3½ times the area of the 130° portion. Therefore, if all of the light from a source is concentrated in the 130° angle, the average intensity over this angle would be 3½ times the mean spherical candlepower. Since the efficiency of the best modern headpiece is 80 per cent, the average intensity in the headpiece over the 130° angle will be 80 per cent of 3½, or 2.8 times the mean spherical candlepower. The maximum beam candlepower can be 5 times the average or 5 × 2.8, or 14 times the mean spherical candlepower.

The present specified maximum of 5 times the average for the maximum beam candlepower of the headpiece of an electric cap lamp is an arbitrary figure and we do not know just whether it is the best value for all conditions underground or not. Some of the metal miners feel that the value should be considerably greater and, in fact, go so far as to recommend a polished reflector.

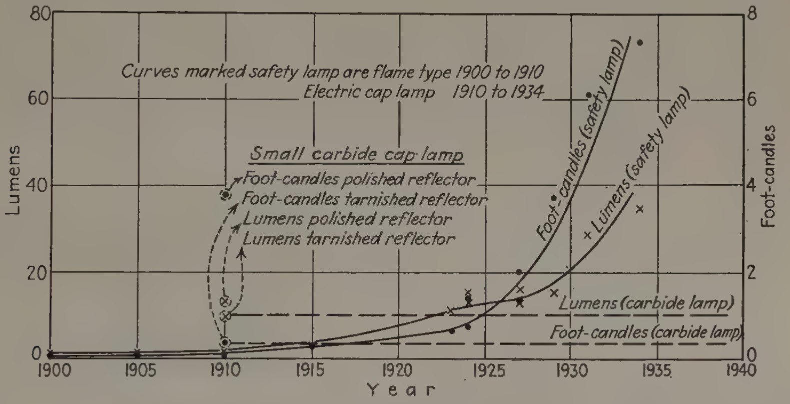


FIG. 7.—CURVES SHOWING PROGRESS IN DEVELOPMENT OF FOOT-CANDLES AND LUMENS AVAILABLE TO THE AMERICAN MINER 1900 TO 1934.

Trend in Underground Illumination

The actual trend in illumination values available for the underground worker in America from 1900 to 1934 is shown in Fig. 7 and Table 2. On this curve are plotted both foot-candles and lumens, showing that the foot-candle values have increased at a more rapid rate from 1923 than have the lumen values. This is due to the use of more efficient reflectors and a better light distribution as permitted by the revised schedule of the U. S. Bureau of Mines. The maximum actual illumination in foot-candles today is 50 times what it was in 1910. The lumen value today is 20 times as great as it was in 1910.

The foot-candle curve for the electric cap lamp passed that of the open-flame lamp in 1917, while the lumen curve of the open-flame lamp was passed in 1925. The foot-candle values of any lamp with a polished reflector is of little importance since the illuminated area is too small to be considered a working area.

TABLE 2.—*Illumination Available to the Underground Worker in America, 1900 to 1934*

Date	Type of Lamp	Maximum Beam Candlepower	Foot-candles on Work[a]	Lumens (Average)
1900	Flame safety	0.5	0.014	1.55
1910	Flame safety	0.5	0.014	1.55
	Carbide cap	25 to 100 polished reflector	2.77	13.5
	Carbide cap	3.5 tarnished reflector	0.38	10
1915	Electric cap	2.2	0.245	3.6
1923	Electric cap	5.5	0.612	11
1924	Electric cap	6.5	0.722	13
	Electric cap	15	1.67	19.5
1927	Electric cap	12	1.33	13
	Electric cap	15	1.67	19.5
	Electric cap	18	2	16
1929	Electric cap	34	3.78	15.1
1931	Electric cap	26	2.78	12
	Electric cap	34	3.78	15.1
	Electric cap	55	6.1	28.7
1934	Electric cap	66	7.32	34.2

[a] Based upon hand lamp 6 ft. from work or cap lamp 3 ft. from work.

It is never safe to predict what the future will bring forth by extending such curves as those in Fig. 7. Although, as shown on the curve, we have passed minimum values recommended for industrial lighting, further increases can be expected. The actual maximum will no doubt be largely governed by the question of glare. There seems to be no reason why the maximum beam candlepower should not go as high as 10 times the average if objectionable glare does not take place.

Special Lamps

There are some special applications of battery-type lamps for underground service that require some modifications of the regular electric cap lamp or the use of a hand lamp. Some of these require a polished reflector and others a matte surface reflector. The following are some of the special permissible lamps at present in use:

1. Hand lamp with cap-lamp headpiece mounted on battery, polished or matte surface reflector.
2. Officials lamp. This is a cap lamp with button switch on the battery and is lighted by pressing the button, which must be held in closed position.

3. Trip lamp. Uses a red lens. For danger signal on end of trips of cars.

4. Mule or horse lamp.

5. Permissible flood lamp. Made up using a number of cap-lamp batteries. Can use polished or matte surface reflector.

A rather novel idea in regard to the effects of good and poor lighting was expressed by Dr. M. Luckeish and Frank M. Moss, of the Research Lighting Laboratories of the General Electric Co., in a recent publication, "Lighting for Seeing." A curve from that publication is reproduced in Fig. 8. In drawing up this curve, it is assumed that a human being is a machine whose output is divided between useful and useless work and in normal work he does not expend the maximum amount of work of which he is capable. The entire height or ordinate of the curve represents the total amount of energy a man has to expend. The horizontal distance or abscissa represent the intensity of illumination. The length of the ordinate from the base to the lower curve represents the energy actually required for useful work. The length of the ordinate between the lower and upper curve is the energy wasted in useless work due largely to insufficient light and can be called nervous energy used up in *trying to see.* The length of the ordinate from the upper curve to the top is the unexpended energy and can be called the reserve or safety factor.

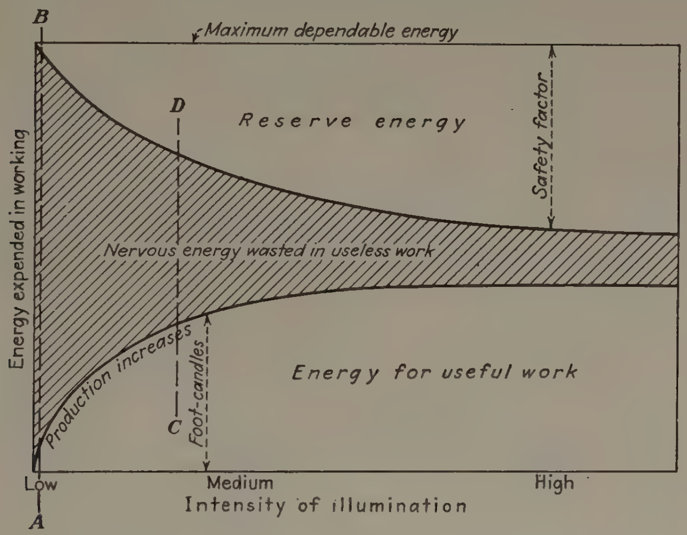


FIG. 8.—EFFECTS OF GOOD AND POOR LIGHTING. (*Luckeish and Moss.*)

Applying this curve to the condition in a coal mine, the vertical line AB would represent an illumination of, say, 0.1 foot-candle. Very little useful work can be accomplished while a large amount of energy is expended in trying to see. The reserve energy is very small. Under such conditions, a miner would go home in a very irritated state, if not nearly exhausted.

On the other hand, let us assume that the illumination is such as represented by vertical line *CD*. A comparatively large amount of useful work is performed with less than half the nervous energy, leaving a goodly amount of unexpended or reserve energy. Such a miner will go home feeling well and will be looking around after dinner for some recreation with his family to work off his surplus energy.

The writer firmly believes that the ideas expressed in Fig. 8 are worthy of careful consideration and may account for the greatly improved conditions that have been noticed at various coal mines when the newer types of electric cap lamps have been installed.

DISCUSSION

(*Harry E. Nold presiding*)

H. E. Nold,* Columbus, Ohio.—A freshman in Ohio State University, who had been a loader in coal mines for two years, during an interview picked up an electric cap lamp and said that the miners would not allow its use because it did not give warning of bad air. I told him bad air was an indictment of management in that proper ventilation was lacking. Miners should not have to do without good lights because of poor management.

T. G. Fear,† Pittsburgh, Pa.—The development of the electric storage-battery cap lamp is one of the greatest improvements in mining in the past 15 years. At a bituminous coal mine producing 4000 tons per day, there was a slate band in the coal seam, which the miners discarded while loading the coal. When the latest type, model "K" Edison lamps were installed underground there was a reduction of 31 per cent in the amount of refuse taken out on the picking tables in the tipple. This reduction was due entirely to better face preparation, made possible by the increased illumination of the electric cap lamps. Electric cap lamps are being used advantageously in nongaseous mines and have reduced personal injuries, especially to transportation crews. The efficiency of the underground day worker has been increased because of the elimination of the time lost in filling and adjusting the old-style lamps.

W. Reid,‡ Scotland.—In Scotland the coal output is about 2 tons a man-shift, and the cost of changing lamps is much more than in the United States because the production per man-shift in the latter is three times that in the former country. There is, however, a definite tendency to introduce electric cap lamps in Scotland. Our estimated loss of time with carbide lamps is 45 minutes per shift of 7½ hours.

H. N. Eavenson,$^\phi$ Pittsburgh, Pa.—Mr. Bright's paper is of great value. I agree with Mr. Fear that more attention should be paid to lighting the working face. In the Frood nickel mines, Sudbury, Ontario, 1500 cap lamps have been installed at considerable saving in cost when compared with carbide lamps.

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* Professor of Mining Engineering, Ohio State University.

† Assistant to the President, H. C. Frick Coke Co.

‡ Fife Coal Co.

$^\phi$ Consulting Engineer.

R. MAIZE,* Harrisburg, Pa.—As to inspectors objecting to firebosses using electric cap lamps, the State Department of Mines does not object if the right kind is used. The flame safety lamp should not be subordinated to the electric cap lamp; examination for gas by fire bosses is the first consideration, the roof second. With the ordinary electric cap lamp on his head a fireboss could not see to test for a small percentage of gas with a flame safety lamp, but with the new approved cap lamp, which has a button on the battery that must be pressed to get a light and when released extinguishes the light, the fireboss can make a more accurate test for gas.

J. D. COONER,† Scranton, Pa.—We operate 17 mines and firebosses are not allowed to carry the ordinary electric cap lamp, but they are allowed to carry the approved type of flash light.

R. D. HALL,‡ New York, N.Y.—I doubt that nystagmus is caused by defective lighting, as has been suggested. Mine lighting is better today in Great Britain than in the past, and there is more nystagmus. No lighting was probably worse than that in the gassy mines of the anthracite region in this country in the early days, but there was no nystagmus. From this it might be inferred that nystagmus was the effect of better lighting but, as with better lighting it has not exhibited itself here and has not increased with the introduction of the brilliant light of the electric cap lamp in Great Britain, such a conclusion is absurd. It would seem we must look elsewhere for the cause. The explanation may be found in the warmer atmospheres of the British mines, which favor the development of a specific germ, as heat favors the growth of the *ankylostoma*, or hookworm. But to propagate, it must first be introduced. Immunity, therefore, depends on lack of infection and lack of opportunity for germ life.

MEMBER.—I recommend the reading of I. C. F. Statham's articles on Coal Face Lighting by Power in the *Colliery Guardian* for July and August, 1934, in which are described the Manor and Coulson fixed electric system in which the wiring is in conduit and each bulb is protected by four guards. The bulb was developed by the Safety in Mines Research Board of Great Britain. Also, there are compressed-air protected lighting systems, lighting by local induction, and a turbine lamp.

G. BRIGHT (written discussion).—In reply to Mr. Nold's remarks that one of his students stated that miners in a certain section would not allow the use of electric cap lamps since carbide lamps gave warning of bad air, this idea is a fallacy, because a carbide lamp will not go out until the oxygen content of the air goes down to about 11 per cent. A person starts to feel uncomfortable when the oxygen content is reduced to 16 per cent and he will succumb when the content reaches about 12 per cent. In other words, the man and the carbide lamp will go out at about the same content of oxygen, so that the carbide lamp cannot very well be used as an indication of a deficiency in oxygen.

Mr. Fear has had such a wide experience in the use of various types of illumination underground that his opinions are extremely valuable, because they are based upon actual facts, not upon theory.

Since Scotland seems to be the pioneer in Great Britain regarding the use of the electric cap lamp, it will be interesting to watch the statistics to see whether the adoption of the cap lamp in Scotland will result in a marked reduction in nystagmus.

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* Department of Mines, Pennsylvania.

† Hudson Coal Co.

‡ Engineering Editor, *Coal Age*.

Mr. Eavenson's remarks regarding the electric cap lamps at the Frood mine of the International Nickel Co. are an indication of what the metal mines think of the modern electric cap lamp. A number of the large metal (gold) mines in northern Ontario and Quebec are adopting electric cap lamps.

In regard to Mr. Hall's remarks about nystagmus, it is true that there is not an agreement among the coal miners and the physicians in England as to the real cause of nystagmus. No doubt the early anthracite miners had poor light but they had not been working under poor lighting conditions for several generations, as is true now in England. If a warmer atmosphere underground is responsible for nystagmus, then why is this disease not prevalent among the metal miners in our western mines, or among the gold miners of South Africa, where high temperatures are almost unbearable?

For years the British mine operators and British physicians felt that poor lighting was not the cause of nystagmus. During the past two years, however, there has been some change of opinion in Great Britain, and many now feel that poor lighting has played a prominent part in the cause of nystagmus. In some of the few instances where better lighting has been installed there has been a marked decrease in nystagmus. Mr. Hall's statement that mine lighting is better today in Great Britain applies to a very small percentage of the mines. The total number of safety lamps used in England in 1933 was 661,689. The total number of flame safety lamps in use in 1933 was 270,000, and of these about 90 per cent were of the older type, giving an average mean horizontal candlepower of about ½. The total number of electric hand lamps in Great Britain in 1933 was 362,764. Of these, over 95 per cent were of the single-cell type giving not over one mean horizontal candlepower. With less than 10 per cent of the lamps in use meeting the new lighting regulations recently put out by the British Mines Department, it can be readily seen that the surface has been barely scratched regarding adequate underground lighting in the mines of Great Britain.

I am familiar with the articles on Coal Face Lighting in the *Colliery Guardian* of July and August, 1934. The systems described are not only complicated but very expensive to install and very expensive to maintain. These systems could not, from an economical standpoint, be installed in America on a commercial scale.

Recording of Roof Subsidence

By H. Landsberg*

(New York Meeting, February, 1936)

Subsidence caused by mining operations has been a matter of interest for the mining engineer for just 111 years, since the Belgian committee for study of subsidence in the city of Liége submitted its first report in 1825. This interest is fairly expressed in the enormous amount of literature that has since appeared on this subject. A bibliography, probably far from being complete, which preceded this study as a preliminary survey, comprises more than 800 titles. In spite of this fact the problem remains today as acute as it has ever been. The questions of as good a recovery as possible of the national resources, the safety of the operations, and the protection of private and public property are the important angles from which to look at the problem.

The Problem

To systematize an investigation regarding the effects of mine excavations two different aspects have to be dealt with: (1) the subsidence of the immediate roof above the workings, affecting the safety of the workers and the progress and efficiency of the operations; (2) the subsidence of the surface, which is also, from the point of time, the second event. The immense importance of surface subsidence as regards the impairment of surface structures of every kind in a mining area is self-evident. Both effects result from reactions of the strata overlying the excavations and they depend mainly on the extent of the excavations and the structure and geological condition of the strata. It has been stressed repeatedly, in this country principally by G. S. Rice[1],[†] that for the understanding of the phenomenon of subsidence accurate surveys are the most urgent necessity. In the past, however, at least in the normal course of mining operations, these surveys have been very crude. The control of the roof in the mine is left to the observation of the behavior of the timber, warning cracks and sounds in roof and walls and to the general judgment and experience of the worker and his supervisors. In any given case this may or may not be sufficient to prevent fatalities but certainly it does not give any accurate information as to what is happening in roof and strata. On the surface, principally in endangered areas, surveys have been conducted more frequently, but also in this case usually weeks or even

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† Numbers refer to references at the end of the paper.

months elapse between the single observations, so that at the best the effects can be stated accurately although the rate of change remains still obscure. Broadly speaking, this means that underground there prevails a very subjective method of personal experience and on the surface a defective set of measurements. If one is permitted to use an analogy, this would be the same as a weather bureau describing the air temperature by highly individual terms such as "warm" or "cold," in the one instance, and making a thermometer reading every month, in the other. The aim of further study of the subsidence phenomena, therefore, must be the incessant record underground and on the surface, to get complete information of what is going on. The problem of continuous surface records is still in a state of planning so far as our own work is concerned, but it is hoped something in this line may materialize within a short time.

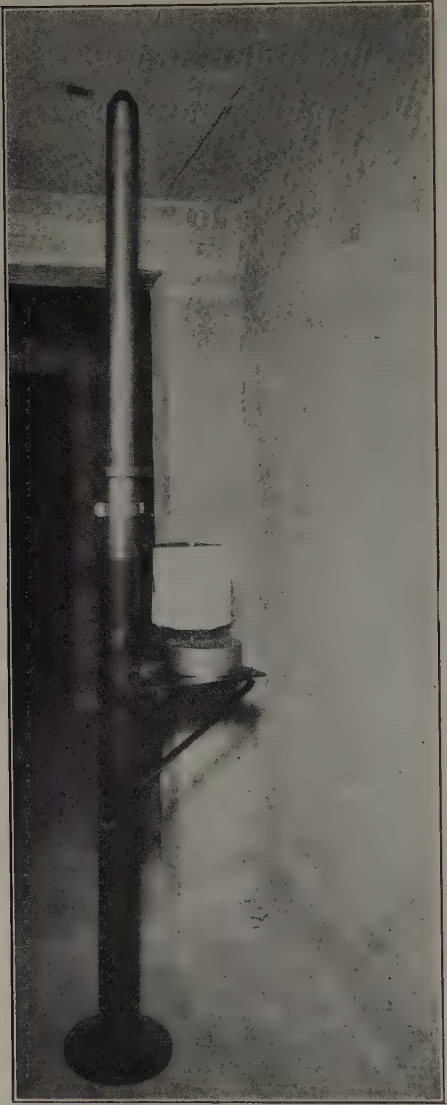

FIG. 1.—CONVERGENCE RECORDER.

CONVERGENCE RECORDS

The present report will deal principally with underground records of roof settlement. As implied in the previous statements, we cannot rely on occasional survey observations or personal impressions, but need incessant records. A method to achieve this was first developed by the British Safety in Mines Research Board as reported by A. Winstanley[2]. The self-registering instrument in use is the so-called convergence recorder, which has been very helpful in collecting information on roof settlement, which was valuable for control and development of operations in longwall workings. The results obtained in British mines in the five years since the instrument has been in use are described by Winstanley[2], Barraclough[3] and Faulkner[4,5]. (See discussion by R. V. Wheeler, p. 147.)

The mining system used in the United States is still mainly room and pillar work, therefore observations of subsidence under these conditions were necessary. G. S. Rice[6] suggested the use of the convergence recorder, and the present report deals with what, to our knowledge, are the first results of that type obtained in this country. The recorder used, although in principle the same as the English type, has been simplified in construction and was built to meet exceptionally bad conditions, so that the instrument in itself would be able to withstand rough handling and occasional small roof falls. The instrument (Fig. 1) consists of two steel tubes with a sliding fit. The upper one is pressed by a spring against the mine roof and has a recording pen attached at one side, and the lower one stands on the floor and bears a recording clock and drum. Any change in distance between roof and bottom will be marked on the record. This does not discriminate between roof and bottom movement, which can be done only by more complicated instruments; it gives, however, a chance for a first general survey.

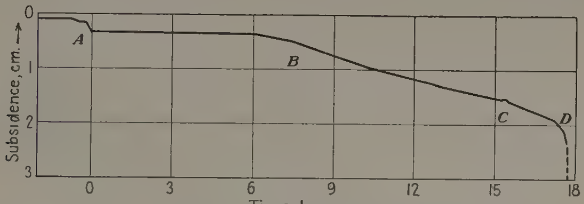

FIG. 2.—RECORD OF CONVERGENCE RECORDER.

One instrument has been in constant use in mines of the Broad Top coal field of central Pennsylvania since February, 1935. The records of this instrument will be discussed, just to give one example of practical results obtained. It is not possible to generalize the findings for the time being, and many more records under all types of conditions are needed and urgently suggested. Geological and general conditions in the Broad Top field are given in the publications of J. H. Gardner[7] and J. D. Sisler[8]. The first experiments were conducted in the No. 9 mine of the Rockhill Coal and Iron Co. near Wood, Pa. The seam is somewhat less than 5 ft. thick and the overlying material is about 250 ft.; the immediate roof consists of 4 ft. of a soft shale with 24 ft. of shale and thin sandstone above.* The instrument was installed approximately 8 ft. from a breakline and the last pillar was being taken out. It could be seen from the timbers that the roof pressure was considerable and that the roof generally was in bad condition. Fig. 2 gives the final record and shows at the end a cave-in. Point *A* marks the sudden start of roof movement, which took place after the pillar had been taken out; *B* shows an abrupt increase in the rate of subsidence, *C* another little jump and *D* the cave-in. The strong construction of the instrument proved here to be of particular value, as it could be recovered without having been damaged. The movement started 18 hr. and 20 min. before the cave-in and corresponds, at least in the order of magnitude, to a case reported by Barraclough[3] in longwall workings, in which he observed an increased rate of subsidence 26 hr. before the fall. Excluding the record of the last 20 min. before the fall, which shows an extremely large rate, the rates of subsidence are shown in Table 1.

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* The room-and-pillar mining system is employed, and 40 per cent of the coal is taken out in the first mining. The pillars are withdrawn, the breakline follows the line of work. The rooms are timbered and the posts are regularly distributed in the rooms.

TABLE 1.—*Rates of Subsidence*

	CM.	IN.
Total rate in 18 hr	1.77	0.70
Corresponding rate for 24 hr	2.36	0.93
Corresponding rate for 7 days	16.52	6.50

Some of Barraclough's[3] longwall observations give a weekly rate of 12.8 in., of which 3 in. were in the last 24 hr. before a roof fall. Again, some agreement in the order of magnitude is shown, although a strict comparison is impossible because of the entirely different conditions.

TABLE 2.—*Weekly Rates of Subsidence*

Week	Face Distance, Ft.	Cm. $\frac{dx}{dt}$	In.
1	150	0.17	0.067
2	135	0.31	0.126
3	125	0.43	0.169
4	115	0.56	0.220 pillar out!
5	105	0.14	0.055
6	95	0.17	0.067
7	80	0.19	0.075
8	65	0.24	0.098
9	45	0.29	0.114
10	35	0.45	0.177
11	25	0.60	0.236
12	15	0.90	0.354

The instrument was then removed to the No. 1 mine of the same company at Robertsdale, Pa. and placed in a gob of room 22 of their development 26th South. The general situation was approximately the same, although the immediate shale roof seemed to be better than in the other mine. The convergence recorder remained there for just three months and during that time the face of the pillar retreat line changed its distance from 150 ft. to 5 ft. The records show slow and steady movements and sudden jumps side by side. The sudden movements seem to occur as in the Broad Top field, when a pillar is completely removed. Other influences that may be expected to produce such jerks will be discussed a little later. From the reports on longwall roof control, we gather that the rate of subsidence increases suddenly during undercutting of the coal[2–5].

Table 2 gives the weekly rates of subsidence in our tests in relation to face advance, or, expressed mathematically, height reduction with time (dx/dt).

If we plot these data as shown in Fig. 3, we see two distinct cycles, which correspond in their rates very closely. This may be interpreted as pressure cycles, influenced by the advance of the workings and at the locality of the instrument apparently effecting a relief of pressure after the first pillar extraction. We refer in this connection to the corresponding results of J. R. Dinsdale[9]. The rates and their increase are not exactly uniform, and this may probably be attributed to the rate of face advance in different periods, as the particular mine was working sometimes longer and sometimes shorter intervals at a time. Faulkner[5] has shown that not only the actual distance from the face but also the rate of change of this distance has a large influence on the speed of settlement.

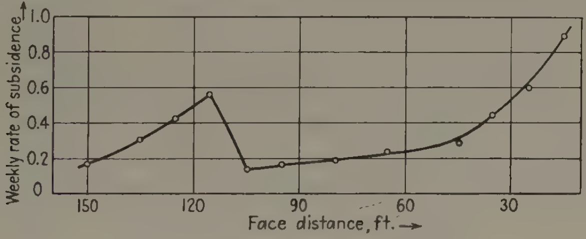

FIG. 3.—PLOT OF DATA OF TABLE 2.

THEORETICAL CONSIDERATIONS

An attempt has been made to analyze the rate of subsidence a little more closely in order to get a better idea of the mechanism that underlies subsidence. It is fully realized that the following derivations can be only preliminary, and a full explanation will need a great many more observations under numerous variations of conditions. The inspection of the second part of the rate curve plotted versus the face advance shows a more or less regular increase of the rate. The observations plotted on double logarithmic paper can be represented by a straight line. This means that $\log r = a + b \log y$, where r represents the rate of subsidence $\frac{dx}{dt}$ and y represents the face distance. This equation can be written as:

$$r = ay^b$$

and in this particular instance this would be $r = \sqrt[1.05]{\frac{13.72}{y}}$ which means that the closer the face approaches our measuring point the higher the rate of subsidence becomes, as would be expected.

Theoretical explanations of the whole phenomenon would have to be mainly concerned with the determination of the meaning of the constants if we take for granted a logarithmic law for the increase of subsidence with face advance. These constants certainly imply two things: (1) the overburden, represented as the depth of the workings (which stands for weight or pressure) and (2) the physical properties of the material. Up to date there are different conceptions as limiting cases for subsidence of the mine roof, such as elastic bending and plastic flow. Elastic bending has been discussed by Young and Stoek[10] and Briggs[11] under the idealizing assumption of a single prismatic beam spanning the excavation. The aim of their computations was to determine the occurrence of breaks when the elastic limit is exceeded. For our present observations, it was of interest to see whether the theory of elastic bending could explain materially the amount of observed roof motion.

Assuming a prismatic beam, subject to elastic bending, there are two subdivisions: the cantilever fixed at one end and the beam supported at both ends. Probably no single one of these ideal conceptions is fulfilled, but if the equation of elastic deflection for these two cases is solved the maximum deflection gives an equation of the form:

$$D = a\frac{Wl^3}{EI}$$

where W is the total uniform load of the beam, l the span, E the coefficient of elasticity. I represents the moment of inertia, which is $bd^2/12$ for a rectangular base b and a depth d. The constant a is determined by the assumptions made regarding the support of the beam. Reasonable values may be substituted in this equation to find what overburden would explain the movements observed. Whatever limiting conditions are assumed, for spans within 200 ft., elastic bending of a layer less than 10 ft. thick could explain movements of the observed order of magnitude. The deviations of such a theoretically computed movement from the observed is large only for rather small spans.

This procedure was repeated for the intimated condition that the roof movement is due to plastic flow of the solid material. The theory of such cases has been developed by Nádai[12] and if we solve again the corresponding equations for the amount of overburden needed to explain the observed motions we find that layers several thousand feet thick would be required. From these theoretical considerations it may be concluded and emphasized that most of the roof motions can be explained by elastic bending of the immediate roof layers, which conforms to the old experience that only these layers usually show breaks and that the higher layers settle later. Plastic deformation, on the other hand, seems to be improbable for our coal mines, working under comparatively shallow cover. In deep mines, like those in South Africa, the observed creeping of the roof may be attributed to plastic flow. What influence inhomogeneous roof material and motion along natural slip lines has on roof subsidence is still an entirely open question.

OUTSIDE FACTORS OF INFLUENCE

Further study of subsidence will have to take into account the other stresses to which the solid rocks are subject, due partly to mining operations and partly to geophysical movements. Vibrations due to blasting will certainly affect the roof, especially if it is already subsiding. Barraclough[3] observed sudden jumps of $\frac{1}{16}$ to $\frac{1}{8}$ in. in his convergence records at the times of shot-firing. Among natural stresses are tidal forces, observed by Schweydar[13] and Tomaschek and Schaffernicht[14]. Dr. P. D. Foote reported recently new measurements at Pittsburgh showing a tide of the solid earth with an amplitude of about one foot[15]. Besides these, the incessant vibrations known as microseisms, although exceedingly small, may have an effect in an already weakened structure. That the vibrations produced by earthquakes are liable to be most harmful in undermined areas has been shown in the earthquake of Nov. 21, 1932, for the western German Ruhr coal district[16]. The effects of earthquakes in relation to mine disasters have not yet shown clearcut relationships[17,18] although the coal district of Upper Silesia has found it advisable to establish six seismograph stations to record earthquakes and "bumps"

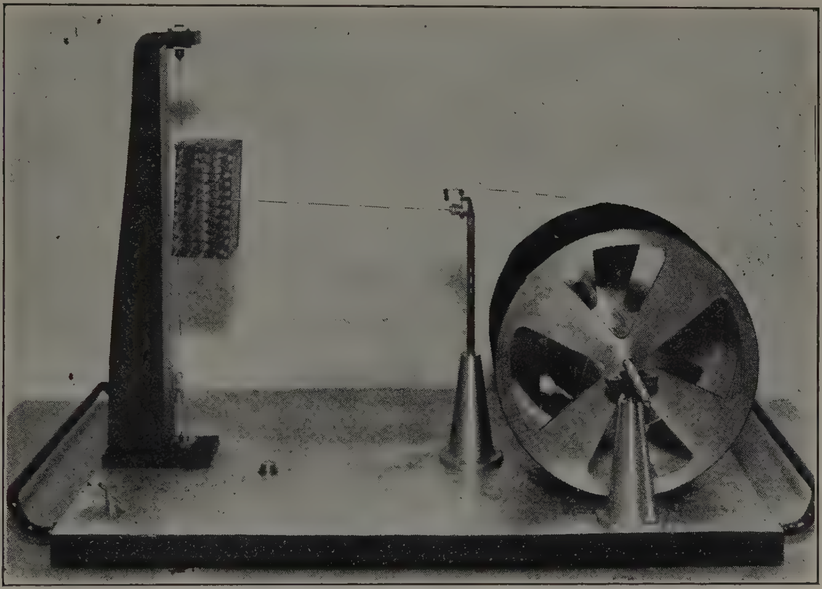

FIG. 4.—PORTABLE MECHANICALLY RECORDING SEISMOGRAPH.

in this region. A survey for seven years with recordings of 137 vibrations and 87 bumps reported by Rellensmann[19] shows, for instance, a more frequent occurrence of these in winter than during the summer; also, the distribution of daily time indicates preferred hours at which those events happened. This meets with certain beliefs of experienced miners, which have been treated in the past largely as superstition. Organizations in other coal fields similar to the one established in the Upper Silesian field seem to be necessary to check upon their stories.

For such purposes a special, portable, mechanically recording seismograph unit has been developed (Fig. 4). Except the steady mass, everything is made of aluminum. The construction is that of a boundary case of a bifilar vibrograph, which avoids the delicate blade springs of the Spindler and Hoyer type of the Wiechert seismograph. Most other transportable models could probably not be used underground, mainly because their photographic recorders are too complicated to handle under these circumstances. Results obtained with the vibrograph cannot yet be reported.

Such geophysical methods, however, seem to be the only means of getting any information of what is happening in the layers between seam and surface. The events in these inaccessible parts are just the ones that determine the chronology and extent of underground and surface subsidence.

ACKNOWLEDGMENTS

The writer wishes to express his thanks to Dr. L. E. Young, of the Pittsburgh Coal Co., for helpful discussions and continued reference to publications. He is also indebted to Messrs. G. S. Rice, Washington, D.C., and A. A. Archer, Pittsburgh, for suggestions.

Particular thanks are extended to Mr. J. George, Superintendent, and Mr. E. Corrie, Mining Engineer, of the Rockhill Coal and Iron Co., Robertsdale, Pa., as well as to the foremen and workers of this company for their splendid cooperation in taking care of the convergence recorder. The Mine Safety Appliances Co., Pittsburgh, was kind enough to build and lend to us a second convergence recorder.

Finally, appreciation is expressed to members of the School of Mineral Industries staff for their courtesy in reading and criticizing the manuscript.

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DISCUSSION

(*George S. Rice presiding*)

R. V. Wheeler,* Sheffield, England (written discussion).—Dr. Landsberg refers to a convergence recorder developed by the Safety in Mines Research Board, and correctly says that the first design was by A. Winstanley. This design was too complicated, however, and the instrument actually used for the work reported by Winstanley, Barraclough and Faulner was designed by H. Lloyd.

H. P. Greenwald,† Pittsburgh, Pa. (written discussion).—The convergence study in this paper by Dr. Landsberg should be of interest to all coal-mining men who desire knowledge of roof action based on records made by instruments rather than on irregular observation and personal experience. The first such study made on this side of the Atlantic was reported to the Institute in 1934 by T. L. McCall[20]. Measurements of lateral movement and convergence were made on large pillars being extracted by retreating longwall workings. Dr. Landsberg's simplification of the British convergence recorder has made it more rugged, but at the same time brings in questions of accuracy. The instrument (Fig. 1) has a broad base resting on the floor and a partly pointed end pressed against the roof. This construction assumes that there will be no superficial movement or disintegration of the surface of either roof or floor; such a condition will not obtain in many cases. An alternate procedure is to drill holes in roof and floor in which pins can be fastened in such manner that only mass movement of strata will be recorded when the instrument is placed between the pins. Use of such pins appears imperative in every case in which measurements are to be accurate to the nearest 0.01 inch.

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† Senior Physicist, U.S. Bureau of Mines, Pittsburgh Experiment Station.

[20] T. L. McCall: Further Notes on Bumps in No. 2 Mine, Springhill, Nova Scotia. *Trans.* A.I.M.E. (1934) **108**, 41–60.

The graph of Fig. 2 is important because it shows movements many hours before a fall. Such movement will not be noticed by those working in the vicinity unless it is accompanied by the noise of breaking strata or unusual behavior of timber. The writer has seen similar records from another investigation now in progress, details of which are not yet ready for publication.

A better interpretation of the graph of Fig. 3 would be possible if complete data were available on the mining operations and any falls that occurred while the recorder was in operation. Obviously there is a direct connection between these events and the records obtained. In all probability, full value of the records can be brought out only with complete data at hand.

With no intention of being specifically critical, it seems wise to put in a word of caution against too literal an acceptance of conclusions based on theoretical consideration of the evidence presented. Taking the industry throughout one field as a whole, conditions vary so widely from mine to mine that many studies must be made before conclusions drawn therefrom can be applied to other conditions with some degree of certainty.

Whether or not the outside factors of influence noted in the concluding paragraphs of the paper are important in mining in this country may develop from future investigations. The method proposed is interesting and, unfortunately, one may expect much work and possibly some difficulty in bringing it to perfection. Because of the scope of the entire problem of roof movement in mines, it is to be hoped that Dr. Landsberg can expand his work and that similar investigations can be undertaken in widely separated mining districts.

L. E. YOUNG,* Pittsburgh, Pa.—Our interest in the portable seismograph arose out of a desire to anticipate serious roof movements at the working face. We have a stepped retreating face in a bituminous coal mine in the Pittsburgh district where the 15-ft. rooms, 300 ft. deep, are on 78-ft. centers. The break line is approximately at 45° from the entry. The coal is 5 to 6 ft. thick. The draw-slate overlying the coal is held on posts and crossbars in the advancing work. The pillars are drawn by a series of cut-overs and slabs, and falls are made periodically as local conditions warrant; obviously the mining cost will be lowest when the percentage of coal recovered in slabs is largest. It is important that the working places be maintained in a safe condition, and, with the regular operation of the mine and multiple shifting, it is possible to remove the pillars very rapidly.

It is hoped that by the use of a portable seismograph it may be possible to discover what vibrations or tremors precede a major movement of the area supported on timber and coal fenders. If such phenomena can be observed and their significance determined, it should be possible to schedule the cutting and loading of the coal, the pulling of the timbers, and the making of falls so that there will be no hazard to face men, and the maximum tonnage of merchantable coarse coal will be recovered per man employed.

As Mr. Greenwald may tell you, we are taking advantage of the staff of the U.S. Bureau of Mines Experimental Mine, which is near by, to study the ground movement following the mining of the coal. The purpose of the observations is to try to discover what phenomena are related to the general movement and to the local falls, and, if possible, to work out a mining schedule that will mean safe and economical operation.

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* Vice President, Pittsburgh Coal Co.

MEMBER.—Have you any results so far?

L. E. YOUNG.—No.

H. LANDSBERG (written discussion).—I fully agree with Mr. Greenwald that the data obtained cannot be termed more than preliminary. With regard to the influence of poor roof or floor on our records, it has to be stated that we had the instruments always set on the closest solid layer in order to have a fixed point of reference. Method of fixing pins in roof and floor, however, seems to be more advantageous and has also been used by A. Ullrich [*Glückauf* (1936), **72,** 81-87].

With regard to the earthquakes, I am not sure myself whether they have any influence or not. At least we do not have any evidence of it as yet. However, I do want to say a word of caution. Of course there is motion due to earthquakes and in an already weakened structure it may be an additional factor for a break but is not necessarily the reason for it.

Whether the seismograph is the solution of Dr. Young's problem has still to be shown. If breaks occur in the upper strata before the roof fall happens the seismograph should be able to pick up the resulting vibrations.

Importance of Pulp Density, Particle Size and Feed Regulation in Flotation of Coal

By John T. Crawford,* Member A.I.M.E.

(New York Meeting, February, 1936)

Much has been written of late regarding the flotation of coal as a means of reclaiming the valuable portion of the fines encountered in nearly all methods of coal preparation. Whether the process be wet or dry, whether the dust be sent through the entire process or removed prior to the cleaning by some such method as screening or aspiration, one of the final products consists of a considerable proportion of −48-mesh material. The percentage of this −48-mesh material depends upon the mining method, transportation, storage, and method of preparation. The tonnage of this dust material ranges from 1 to 10 per cent of the total output of the cleaning plants.

Four methods of disposal are available for this coal:

1. Disposal with the refuse from the plant.
2. Addition, after being reclaimed from the wash water, or dust collectors, without cleaning to the larger sizes.
3. Separate disposal as a pulverized fuel.
4. Cleaning so that addition to the larger sizes benefits the product as a whole.

It is also true that flotation in one form or another is the only known way to effectively clean this material. While some processes do a reasonable cleaning job down to and including +100 mesh, by far the greater majority stop at or short of the +48-mesh size.

Much of the literature on coal flotation passes by the feed pulp density and size by merely stating that the feed varies from 20 to 30 per cent solids and that the size range is from −¼ in. or −⅛ in. to 0. Small wonder then that some operators with entirely different conditions, and not knowing the machine capacity, throw up their hands in despair when flotation is mentioned. Too much parallel is drawn between flotation methods as applied to ores and as applied to coal, to the detriment of the latter.

In the flotation of metalliferous ores the feed pulp density ranges from 20 to 30 per cent solids, depending upon the amount of mineral to be floated and upon the size analysis. Thus where 2 to 5 per cent of the

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total is floated, the feed density may range near the 30 per cent solids mark. In floating from 15 to 25 per cent of the total feed, the pulp density will range more nearly at the 20 per cent solids mark for the rougher cells. In the cleaner cells, where upwards of 50 per cent of the tonnage is floated, the pulp density ranges from 15 to 20 per cent solids. In addition, many circuits return middlings to the primary feed cells, which further dilute the feed. In general, dilution increases with decrease in size and increase of floatable mineral content.

In the flotation of coal, the feed is a mineral more highly concentrated than that in ore-concentration plants. Not only are we dealing with a prospective process where from 50 to 90 per cent of the feed is floated but with a material that floats so easily that much of the undesirable material is mechanically entrained, particularly when we know that as a general rule in most coal sludges the finer portions contain concentrations of the particles that should be eliminated. Table 1 gives the size and ash characteristics of several washery sludges derived as the underflow from a thickener reclaiming wash water following a wet process. Other washery sludges, more particularly those on the European continent, contain considerable amounts of ⅛-in. to 48-mesh material.

Table 1.—*Size and Ash Analysis of Washery Sludge*

Head Ash	+48 Mesh		48 to 100 Mesh		100 to 200 Mesh		−200 Mesh	
	Size, Per Cent	Ash, Per Cent	Size, Per Cent	Ash, Per Cent	Size, Per Cent	Ash, Per Cent	Size, Per Cent	Ash, Per Cent
22.7	4.0	6.0	21.0	11.7	24.0	19.1	51.0	30.3
11.7	0.0		9.0	4.6	21.0	5.8	70.0	14.4
11.9	0.8	6.9	17.4	4.7	26.8	7.2	55.0	16.4
13.4	1.0	6.5	16.5	5.8	24.5	8.2	58.0	18.0

Very seldom is any attempt made to float the +28-mesh particles, as it merely results in the bringing over of the slate, clay, pyrite, marcasite, gypsum and calcite particles in the −200-mesh portion, which contains the bulk of the impurities that should and necessarily must be rejected in the tailings if the process is to be successful.

Efficient screening or classification to eliminate the +48-mesh material, which, if the primary washing in the main plant has been done properly, needs no further washing prior to flotation, is either economically impractical or incompletely accomplished. Therefore operators presumably feed the −⅛-in. to 0 material to the flotation units, float off the −48-mesh coal and screen the +48-mesh coal out of the tailings, which is easily done because the bulk of the −200-mesh material has been previously floated off.

As this ⅛-in. to 48-mesh coal varies from 10 to 50 per cent of the total feed at 20 to 30 per cent solids (as reported) and does not enter into the actual flotation conditions, inasmuch as no attempt is made to recover it in the froth concentrate, the actual density in the cells is considerably less than the 20 or 30 per cent reported. In addition to this, little or no mention is made of the dilution caused by return of middlings froth to the cells after addition of water sprays have made the froth transportable. (It should be known that the froth resulting from coal flotation is particularly hard to break down for subsequent handling.)

It has been the experience of the author, in attempting to balance many circuits, that overly large amounts of water entered into circuits and caused serious dilution, which either was not reported as such or was entirely overlooked. One such example is cited here. It deals with a test run on two 43-in. mechanical subaeration cells, made especially for coal flotation. The feed, consisting of coal sludge containing 4.0 per cent of +48-mesh material, entered a 1300-gal. conditioning tank, from whence it was fed to No. 2 cell, which made a final tailing and a rougher concentrate that was fed to No. 1 cell, which made a finished concentrate and a middling that entered No. 2 cell with the primary feed. The results as reported were as follows:

	Ash, Per Cent	Solids, Per Cent	Dry Coal, Tons per Hour
Primary feed	22.7	27.7	2.53
Concentrate	7.8	20.9	1.00
Tailings	58.9	5.5	

As samples were all taken correctly and run by the laboratory, the ash samples must be assumed as correct; also, the percentage of solids in the concentrates and tailings. The concentrate tonnage is reasonably correct because this figure was obtained from the actual output of the filter handling only the concentrate. Calculation of recovery based on the ash analysis gives 70.8 per cent. Calculation of the feed tonnage based on a 70.8 recovery and 1.00 tons of concentrate actually recovered from the filter gives an actual feed tonnage of 1.4 tons per hour as against 2.53 tons reported. This finally was explained as an inexperienced operator's error either in measuring the density of the feed or the gallons per minute of feed, or both. In this particular instance the author was able to decide where the error probably occurred. In many other cases he was unable to do so, generally because of insufficient data. The following data are now available for calculation of this circuit:

	Ash, Per Cent	Solids, Per Cent	Dry Coal, Tons per Hour
Primary feed	22.7		1.4
Final concentrate	7.8	20.9	1.00
Final tailings	58.9	5.5	0.41

With this as a basis (generally most published results contain no more) the following tonnages are derived by calculation:

	Ash, Per Cent	Solids, Per Cent	Dry Coal, Tons per Hour	Water, Tons per Hour
Primary feed	22.7		1.4	
Final concentrate	7.8	20.9	1.00	3.78
Final tailings	58.9	5.5	0.41	7.04

No analyses of middlings or primary concentrates being available, a complete balance of the circuit is impossible. It does show that 10.8 tons of water was leaving the circuit each hour. If this all came in with the primary feed the percentage of solids based on a dry feed of 1.4 tons per hour would have been 11.5. Calculation by assuming the reported feed of 2.53 tons per hour to be correct and calculation of the concentrate tonnage from that with the reported percentage of solids of the concentrate only makes it worse. Further inquiry, however, elicited the information that considerable spray water was introduced into the primary concentrate to return it to No. 1 cell, causing further dilution in that cell, and with the return of the cleaner tailings to No. 2 cell with the primary feed certainly diluted that to a considerable extent. Small wonder that mill operators become gray haired trying to equal results apparently achieved in other localities.

The number of particles per cubic unit is by far the most important factor, rather than pulp density. In dealing with coal at 1.35 to 1.45 dry specific gravity in the form of sludge, a great many more particles are present per ton of dry feed, assuming the screening analyses to be the same, than when dealing with ores at greater than 3.0 sp. gr. It simply means that we have at least twice as many particles present, and when we consider that we are floating from 50 to 90 per cent of this total tonnage it is readily seen that a much more crowded condition is present in the cells. Hence the fallacy of attempting to compare pulp dilutions and tonnages in coal and ore flotation, which is an error commonly made.

It is the author's opinion that where the actual condition in the cells is based on from 10 to 12 per cent solids (by weight) the best results are achieved in the flotation of coal. This may be accomplished in several ways, depending on the circuit employed:

1. By dilution of the primary feed.
2. By return of middlings with a low percentage of solids to the primary feed.
3. By stage addition of the feed to the first two or three cells in the circuit. If one-fourth of the primary feed enters the first cell, the tailings from this cell dilute the feed to the second cell and the tailings from the second dilute the feed to the third cell. Even when the feed tonnage is kept the same and the dilution of the feed increased from 20 to 10 per cent solids, much better results may be expected even though the velocity through the cells is doubled. Table 2 shows some typical results obtained in a laboratory batch machine where the density of the feed was varied.

TABLE 2.—*Effect on Tailings of Varying Pulp Density in Laboratory Cell*

Test No.	Solids, Per Cent	Float Time, Min.	pH	Ash Head, Per Cent	Ash Conc., Per Cent	Ash Tail., Per Cent	Recovery, Per Cent
8	3	3	8.1	13.9	8.4	65.4	91
9	6	6	8.1	13.9	8.1	62.4	89
10	10	7	8.1	13.9	8.5	62.8	90
11	12	8	8.1	13.9	8.4	41.3	83
12	20	13	8.1	13.9	8.8	42.3	85
13	25	17	8.1	13.9	9.7	30.8	80

It will be observed that even with a considerably longer treatment time and a much lower percentage of ash tailings the higher densities produced less satisfactory results than the lower densities.

Table 3 shows the results obtained on a 43-in. six-cell mechanical agitation machine adapted for coal flotation. The circuit was the same in all the tests: feed entered No. 1 cell, concentrates were taken from cells

TABLE 3.—*Effect of Varying Density in Mill-size Cells*

Solids in Feed, Per Cent	Tons per Hour	Ash Content, Per Cent		
		Head	Conc.	Tails
24.0	6.3	13.0	10.7	52.0
22.6	6.1	13.3	11.0	61.6
20.3	6.0	12.8	9.9	60.7
19.4	6.4	13.6	9.3	51.5
18.0	7.3	13.0	9.4	71.0
16.9	6.4	13.4	9.1	59.8
15.5	6.0	12.5	8.7	76.5
15.4	6.1	13.0	8.8	69.4
12.9	6.2	14.0	7.3	50.7

1 to 5 as final products, concentrate from No. 6 cell was returned to No. 3 cell. No. 6 cell made a final tailing. A simple circuit was used and no attempt was made at recleaning, because it was desired to determine the effect of pulp dilution. These tests are characteristic of the results obtained on a sludge containing 1.0 per cent of +48-mesh and 58.0 per cent of −200-mesh material.

With these mill runs in view, a series of tests (Table 4) was made in a standard laboratory batch machine, which show that, beyond a doubt, the greater the dilution, the better the cleaning or separation.

TABLE 4.—*Effect on Concentrates of Varying Pulp Densities in Laboratory Cell*

Test No.	Solids, Per Cent	Float Time, Min.	Ash Content, Per Cent		
			Head	Conc.	Tail.
76	3	8	13.1	6.2	73.0
77	6	8	13.1	6.8	72.6
78	8	10	13.1	6.8	75.8
79	10	8	13.1	7.3	78.6
80	13	10	13.1	7.6	82.3
81	16	12	13.1	8.4	70.1
82	18	12	13.1	8.6	80.0
83	21	13	13.1	8.4	80.8
84	23	13	13.1	8.9	78.7
85	25	15	13.1	8.8	74.9
86	28	18	13.1	8.9	70.7

Apparently the best all-around results are obtained with a pulp containing from 11 to 12 per cent solids. Higher densities show gradual increase in ash content of the concentrates, and while better results are obtained at the very low densities, the actual ash reduction resulting is not worth while when consideration of the amount of dilution water necessary is involved.

Particularly is this true when flotation is merely an auxiliary process in a mill where the bulk of the tonnage is cleaned by some other process. Most washing plants operate with a closed water circuit. Addition of a large volume of water to dilute the feed to the flotation cells, and its subsequent handling, is a most important factor. Thickening and/or filtration of the concentrate with the return of the reagent-bearing filtrate to the flotation cells as dilution water is a logical step comparatively easy of accomplishment, but thickening and/or filtration of the tailings is not quite so feasible, as this product would contain practically all of the fine clay entering the feed to the main plant with the raw coal. Filtration of the tailings after thickening may not be economically justifiable, as this is a waste product, so that wastage of the thickened tails or of the entire tailings and water is the only recourse left. Inasmuch as the cost of tailing wastage and pumping the fresh-water supply to the plant depend almost entirely on the amount of water added to the cells, it will depend to a great extent on local conditions as to whether or not it would be advisable to sacrifice metallurgical results or add additional recleaning cells in order to save elsewhere. This is a delicate balance of both local and market factors.

Another series of mill runs was made in a six-cell 43-in. mechanical agitation machine. The primary feed entered No. 4 cell, froth from Nos. 4, 5 and 6 cells was returned to Nos. 1 and 2 cells. No. 6 cell made mill tailings. Nos. 1, 2 and 3 cells made a final concentrate and the tailings from No. 3 cell (middlings) entered No. 4 cell with the primary feed. Typical results are shown in Table 5. Spray water was added to rougher concentrates.

TABLE 5.—*Typical Results in Six-cell Mechanical Agitation Machine*
Tons per Hour, 6.0

	Solids, Per Cent	Ash, Per Cent	Cell No.	Cell Concentrates, Per Cent	
				Solids	Ash
Head	21.5	13.1	1	23.1	6.5
Concentrates	22.0	6.9	2	22.5	6.4
Tailings	3.2	59.4	3	22.8	7.8
Middlings	2.5	41.4	4	22.9	8.6
			5	23.0	9.3
			6	22.8	11.2

If we leave such results as they are we get an entirely erroneous idea of the true conditions. Balancing the circuit makes available the figures in Table 6.

TABLE 6.—*Calculated Figures Obtained from Balancing Circuit of Table* 5

Item	Ash, Per Cent	Solids, Per Cent	Tons per Hour		Recovery, Per Cent
			Solids	Water	
Primary feed	13.1	21.5	6.0	22.0	
Middlings	41.4	2.5	0.5	19.5	
Actual feed to cell No. 4	15.5	13.6	6.5	41.5	
Mill tails	59.4	3.2	0.7	21.2	
Rougher concentrate	9.7	22.9	5.8	19.6	88.0
Cleaner concentrate	6.9	22.8	5.3	18.0	92.0
Spray water				17.2	
Over-all					88.0

In this instance, as in others not cited here, the actual condition existing in the cells is a much lower density than would appear on first examination of the data, owing to addition of spray water and return of middlings to the feed. In this test it was approximately 8 per cent less.

Dispersion of the pulp is obtainable by the addition of such agents as sodium silicate, soda ash, etc., or by use of an acid circuit, which gives better dispersion than an alkaline circuit, which at too high a pH tends to flocculate the particles. Generally, where dispersing agents were used, the tendency was toward a cleaner separation.

Because of the low specific gravity of the coal sludge, and the large number of particles per unit weight, as well as the already high concentration of primary feed, it is absolutely necessary that the rate of the feed to the cells be regulated within as close limits as is possible. Slight variations in percentage of solids or in tons per hour of solids throws any recirculating circuit entirely out of balance and results in poor concentrates, poor tailings, and a heavy build-up of recirculating middlings. Variation in the specific gravity of the solids in the primary feed is so slight as to be negligible but because of the low specific gravity of the solids, and therefore the pulp, automatic density recorders or regulators are practically worthless. The only feasible method is by use of a density flask of at least 2 liters capacity and some means whereby the entire flow of primary feed may be directed into a measuring tank of 50 to 100 gal. capacity. By combining the specific gravity of the pulp with the time interval for filling a tank, the entire procedure for determining feed tonnages takes approximately two minutes.

Where coal dust constitutes the primary or raw feed, the problem is greatly simplified, as specific amounts of dust and water can be measured separately and mixed to obtain almost unvarying tonnages and densities far more accurately than sludge tonnages.

Although under favorable conditions 10-mesh particles of coal may be floated successfully where the size range of the feed is relatively narrow, this does not hold true when much −200-mesh material is presented. It is imperative to keep the percentage of +48-mesh material at a minimum if we are to obtain a clean concentrate and a clean tailing in the −200-mesh size, as it is in this size that most of the cleaning action must take place. Screening of the tailings to recover the coarser coal would seem to be the most logical step if the size range includes much +48-mesh.

In the flotation of coal, the −200-mesh particles pass off first, followed by the 100 to 200-mesh size. Next to float is the 48 to 100-mesh size together with some of the finer pyrite and gangue. Continued flotation to recover the +48-mesh coal results in flotation of the pyrite and gangue. For the particular coal dealt with in this paper flotation continued after an ash content of 75 per cent in the tailings has been reached only tends to float off the pyrite and cause a concentration of it in the concentrate instead of in the tailings. Actually this percentage of ash is not obtainable in full-size mill units, and a 65 per cent ash tailings is more nearly standard practice.

It is possible, however, to obtain an ash of 80 per cent in the rougher cells tailings and the fine pyrites may then be dropped in the cleaner cells. Cleaning of rougher concentrates offers the best solution in evening out the variations due to dilutions, tonnage, and the effect of slimes and dissolved salts in the water, together with offering the best opportunity to drop the pyrite without making low-grade tailings.

Table 7 shows some results obtained with recleaning in a laboratory batch cell. For tests 97 and 99, the heads were 13 per cent solids and cresylic acid was the only reagent used in the amount of 0.52 lb. per ton for the rougher float and 0.28 lb. per ton for each of the two cleaner floats. The sludge used was similar to the last example given in Table 1.

TABLE 7.—*Effect of Cleaning in Laboratory Batch Cell*

Test No.	pH	Head, Per Cent		Concentrate, Per Cent		Tailing, Per Cent		Cumulative Tailing Ash, Per Cent	Remarks
		Ash	Sulfur	Ash	Sulfur	Ash	Sulfur		
97	7.2	13.7	1.20	7.7	1.10	81.6	1.45	81.6	First pass
		7.7	1.10	5.9	1.05	71.6	2.20	76.8	Second pass
		5.9	1.05	5.5	1.00	52.5	1.95	73.7	Third pass
99	7.6	13.7	1.20	8.0	1.05	69.6	1.30	69.6	First pass
		8.0	1.05	5.5	1.00	54.7	1.70	66.3	Second pass
		5.5	1.00	4.6	0.95	48.8	2.10	63.2	Third pass

Table 8 illustrates the difference that may be expected between a single rougher float and a rougher float followed by cleaner cells. In the two examples shown the flow was as follows: For circuit No. 2, the raw feed was added to cell No. 1. The froth over all the cells was taken off as concentrate and cell No. 6 made a final tailings. For circuit No. 4, the raw feed entered cell No. 4, rougher concentrates overflowing cells 4, 5 and 6 were returned as the feed to cells 1 and 2. Cells 1, 2 and 3 made a final concentrate and the middlings or tailings from cell No. 3 entered with the primary feed to cell 4. The only reagent used was 0.40 lb. per ton of cresylic acid. The raw feed was 6 tons per hour.

The previous calculation on circuit No. 4 (Table 6) shows that, owing to the return of spray water with the rougher concentrates and the return of middling with the primary feed, the actual floating conditions in the cells themselves are very different from those in circuit No. 2, so that despite the fact that the actual treatment time is shorter per float, the recleaning circuit gives much better results.

TABLE 8.—*Effect of Recleaning in Mill-size Units*

	Circuit No. 2		Circuit No.4	
	Solids, Per Cent	Ash, Per Cent	Solids, Per Cent	Ash, Per Cent
Heads	16.9	13.4	21.5	13.1
Concentrates	25.0	9.1	22.0	6.9
Tailings	3.2	59.8	3.2	59.4
Middlings			2.5	41.4
Cell No. 1	20.8	9.3	23.1	6.5
Cell No. 2			22.5	6.4
Cell No. 3	23.7	8.8	22.8	7.8
Cell No. 4	25.2	8.9	22.9	8.6
Cell No. 5	27.8	11.0	23.0	9.3
Cell No. 6	29.6	11.2	22.8	11.2

Table 9 gives the size and ash characteristics when using one cleaning float in a laboratory batch cell. For the series of tests shown, 0.53 lb.

TABLE 9.—*Size and Ash Characteristics with One Cleaning Float in Laboratory Batch Cell*

Test No.	Ash	Heads, Per Cent								Rougher Conc. Ash, Per Cent	Rougher Tailings, Per Cent				
		+48 Mesh		48 to 100 Mesh		100 to 200 Mesh		−200 Mesh			Head Ash	+200 Mesh		−200 Mesh	
		Wt.	Ash	Wt.	Ash	Wt.	Ash	Wt.	Ash			Wt.	Ash	Wt.	Ash
113	13.6	1.0	6.1	17.0	5.7	25.5	8.3	56.5	18.2	6.8	81.0	1.4	50.4	98.6	81.6
114	Same as 113									7.5	80.0	2.8	47.1	97.2	81.1
115										7.0	81.8	1.5	48.0	98.5	82.4
116										7.5	77.4	2.5	33.8	97.5	78.6
117										7.7	78.5	1.4	35.4	98.6	80.1
118										7.0	81.7	1.5	36.4	98.5	82.5
119										6.8	79.1	2.4	42.0	97.6	80.1
120										7.0	73.6	5.3	36.4	94.7	75.8
121										7.8	72.1	1.6	55.0	98.4	82.2
122										7.6	78.7	1.5	35.0	98.5	80.4
123										8.2	81.6	1.6	40.2	98.4	82.3
124										6.7	74.7	3.9	30.7	96.1	76.5
		Cleaner Concentrates, Per Cent									Cleaner Tailings Per Cent				
113	5.7	0.8	3.6	14.2	3.5	24.4	4.7	60.6	6.8		61.5	9.5	28.5	90.5	65.1
114	6.0	0.8	3.0	14.4	3.4	24.4	4.7	60.4	7.0		65.8	5.0	23.4	95.0	68.1
115	5.8	0.8	2.9	14.2	3.3	24.2	4.8	60.8	6.7		68.7	5.5	38.8	94.5	70.6
116	6.1	1.0	3.8	13.0	3.5	21.8	4.8	64.2	7.0		65.7	4.9	29.3	94.1	68.4
117	6.3	0.8	3.3	12.6	3.3	22.6	4.7	64.0	7.3		68.5	6.9	42.4	93.1	70.6
118	6.1	0.8	3.8	13.6	3.5	23.4	4.8	62.2	7.0		61.2	7.4	24.1	92.6	64.2
119	6.2	0.8	3.6	12.2	3.4	21.4	4.8	65.6	7.2		68.4	5.9	26.3	94.1	71.1
120	6.0	0.8	3.9	12.4	3.7	21.6	4.8	65.2	7.0		62.4	4.8	23.1	95.2	64.4
121	6.1	0.8	3.4	14.0	3.4	23.8	4.7	61.4	7.0		72.1	6.7	42.8	93.3	74.2
122	6.1	0.8	3.4	14.2	3.3	24.8	4.7	60.2	7.1		75.4	8.3	62.7	91.7	76.7
123	6.4	0.8	3.4	14.0	3.5	26.4	5.0	58.8	7.2		77.4	7.1	41.2	92.9	80.3
124	6.2	0.8	3.3	15.0	3.5	24.6	4.5	59.6	7.1		50.6	8.3	28.6	91.7	53.7

of cresylic acid and 0.23 lb. per ton of kerosene were used in the rougher float and 0.40 lb. per ton of cresylic acid was used in the cleaner float.

Conclusions and Summary

Metalliferous ores and silicate minerals have been found to respond best to the flotation process at a pulp density of approximately 20 to 30 per cent solids. We must assume that this is so because such a dilution permits each particle to move freely with respect to the other particles. Such a separation is necessary in order that the desired particle may be chosen from the group and floated. More dense pulps tend to float the undesirable portions by reason of mechanical inclusion in the froth bubbles rising from the agitated pulp. Coal, by reason of its low specific gravity and friable tendencies, occupies from two to three times more space for the same tonnage at the same pulp dilutions as heavier minerals. Hence to achieve the same looseness of pulp in dispersed condition, a coal pulp must carry a smaller percentage of solids. This fact has been well illustrated in the test runs in both the laboratory and the mill. Uniformly poor results were obtained at pulp densities over 17 per cent solids while the best results were obtained at or near 12 per cent solids.

Regulation of feed tonnage within narrow limits was found to be an important factor in producing satisfactory results.

Where the bulk of the tonnage to be cleaned is less than 200-mesh size, it was found that to float +48-mesh particles in the same pulp was detrimental to the cleaning of the −200-mesh material.

Acknowledgment

The writer takes this opportunity of thanking Messrs. R. E. Zimmerman, M. J. Sayres, S. B. Barley and J. A. Younkins, of the Pittsburgh Coal Co., for their aid in obtaining some of the data included here and for their suggestions during the investigation. Many thanks are due Mr. J. B. Morrow, for his permission to use the data and for his many helpful suggestions during the preparation of this paper.

DISCUSSION

(*Cadwallader Evans, Jr., presiding*)

W. B. Cramer,* Golden, Colo. (written discussion).—One of the very interesting points of Mr. Crawford's paper from the viewpoint of the flotation-of-ore operator is the pointing out by the author of a glaring lack of fundamental knowledge regarding balanced pulp conditions on the part of at least some coal-flotation operators. Accurately determined water balance and process tonnage distributions are the rule rather than the exception in practically all present-day ore-treatment plants and are considered as essential fundamentals embodied in proper plant control.

The author has presented in a clear, convincing manner strong evidence to substantiate his claim that pulp dilution in coal flotation varies considerably from pulp dilution in ore flotation in that in coal flotation the greater the dilution, the better the cleaning or separation. His advocacy of recleaning as further aid to ash elimination parallels plant practice in flotation ore treatment.

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* Associate Professor of Metallurgy, Colorado School of Mines.

H. F. Yancey,* Seattle, Wash. (written discussion).—All coal-mining companies that encounter difficulties in disposing of the finer sizes of coal either as dust or slime will find much valuable information in this report of Mr. Crawford on the cleaning of coal by froth flotation. It is the first investigation based on plant operation that has been published in the United States. All previous reports have been confined to the description of a few isolated plants, or have consisted entirely of studies made with laboratory-size flotation cells. Mr. Crawford's report is based principally on trials with full-size six-cell machines supplemented in some instances by tests in the laboratory. Heretofore all the work in the United States has been done in the laboratory. Several plants have been put in, but the material that has appeared in the literature concerning these plants has been entirely descriptive. There has been no development of technique in the coal-flotation field such as there has been for ore.

The author's conclusions about the effect of pulp dilution agree with studies made by the U.S. Bureau of Mines at the Seattle station. Laboratory tests of a similar feed containing 13.3 per cent ash and 2.96 per cent sulfur at dilutions varying from 1 to 45 per cent solids gave yields of froth ranging from 85 to 94 per cent, which carried from 6.5 to 10.6 per cent ash[1]. The yield and ash content of the cleaned coal increased continuously with increasing concentration of pulp.

The strict limitation of the range of size treated in the flotation operation is stressed by the author, particularly if much material finer than 200 mesh is present. This limitation, of course, is based on the characteristics of the feed used in the work. With more friable coals in which the material finer than 200 mesh is cleaner, it should be possible to treat a wider range of sizes effectively.

In calling attention to specific differences between coal and ore flotation mention is made of the fact that 80 to 90 per cent of the feed must be recovered when coal is treated, as compared with 1 to 10 per cent with ores. The recovery of such a large quantity of material when dealing with coal is often considered more difficult than the recovery of the smaller quantities necessary in ore flotation. From the standpoint of the technical difficulties involved, as distinguished from the practical, I should like to ask if it is not just as difficult to recover 5 per cent and reject 95 per cent, as it is to recover 95 per cent and reject 5 per cent. (That last comment is to arouse some discussion. I think I have much the same views as Mr. Crawford.)

J. T. Crawford (written discussion).—Although sufficient cleaning of a coal sludge may be satisfactorily accomplished in a rougher float only, by use of a very dilute pulp density, recleaning is much to be preferred as the return of the middling from the cleaner cells at a low percentage of solids acts as a diluent to the primary feed entering the rougher cells. The return of the high-ash middlings to the rougher feed is less detrimental than might be supposed.

Any discussions of the relative difficulties of ore and coal flotation are apt to depend upon the individual's point of view. Coal floats readily; in fact, one might well say that coal floats too readily. As Dr. Yancey says, much of the trouble experienced in coal flotation might be eliminated in a coal sludge when the finer (−200-mesh) portions contained less gangue material to be removed than the +200-

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* Supervising Engineer, U.S. Bureau of Mines, Northwest Experiment Station. Presented by permission of the Director, U.S. Bureau of Mines.

[1] H. F. Yancey and J. A. Taylor: Froth Flotation of Coal; Sulphur and Ash Reduction. U.S. Bur. Mines *R.I.* 3263 (1935) 17.

mesh portion. Comparison of coal and ore flotation, it seems, would be more direct, if, instead of comparing feed tonnage, we compared concentrates tonnage.

Coal sludges may and do vary widely in size and ash distribution.

TABLE 10.—*Size and Ash Distribution in Coal Sludge*

Sludge No. 1			Sludge No. 2		
Size	Percentage of Size	Percentage of Ash	Size	Percentage of Size	Percentage of Ash
Head	100.0	16.2	Head	100.0	14.7
+48 mesh	none		+48 mesh	16.5	8.6
48 × 100	5.0	13.5	48 × 100	34.5	7.7
100 × 200	15.0	11.2	100 × 200	21.0	15.2
200 × 325	10.0	7.9	−200 mesh	28.0	29.8
−325 mesh	70.0	18.9			

From the screen analyses in Table 10 it might at first seem that sludge No. 2 would be the most difficult to clean because of the high ash content of the −200-mesh material and the presence of so much +48-mesh material as compared to sludge No. 1, but in reality sludge No. 1 will be the most difficult to treat because of the extremely high percentage of −325-mesh material in it. If a particular cell will handle 1.8 tons per hour of sludge No. 2 giving a 7.3 per cent ash concentrate and a 78.8 per cent ash tailings, it is to be expected that the same cell giving the same performance on sludge No. 1 will not handle more than 1.0 tons (dry) per hour.

Comparison of difficulties, either technical or practical, between coal and ore rougher flotation is hardly practicable. If coal roughers were compared with ore recleaners, both making recovery of the bulk of the cell feed input, it is likely that less difficulty would be shown in regard to coal, for the same screen analyses and when differential flotation is not practiced.

Performance Expectancy of Domestic Underfeed Stokers for Anthracite

By Allen J. Johnson,* Member A.I.M.E.

(New York Meeting, February, 1935)

With a realization of the rapidly increasing importance of automatic stokers as a medium for domestic heating, the Anthracite Institute Laboratory has conducted extensive investigations, over a period of several years, to determine the various outstanding characteristics of stoker design and performance. The purpose of this paper is to summarize these findings as a guide to current practices.

Stokers Investigated

One or more models of the underfeed stokers listed below were included in the various phases of this investigation:

- Beckley
- Burnwell
- Combustioneer
- Electric Furnace Man
- Erie Heat Generator
- Fire Ball (Tests Incomplete.)
- Gilkoal
- Iron Fireman
- Motor Stokor
- Norristown Super-Stoker
- Whitney
- Fuel Saver

Principles of Construction.—The operating principle of the leading makes of underfeed stokers is similar (Fig. 1). A plunger[1] or slowly revolving worm[2], conveys the buckwheat or rice sizes of anthracite to the center of a specially constructed tuyere pot where air is admitted to the coal through suitable ports of varying design. As the fuel burns, it is slowly forced to the outside circumference of the tuyere by the pressure of the incoming coal; eventually the ash falls over the edge of the pot, either entirely by gravity or aided by mechanical sweepers. In at least one model, the entire firepot revolves. While various special details have been included in the design of a number of stokers, they do not affect the general principles as described above, and therefore have not been included in this discussion.

Manuscript received at the office of the Institute Dec. 11, 1934.

* Director, Anthracite Institute Laboratory, Primos, Pa.

[1] Whitney and Norristown Super Stoker.

[2] All other models.

OPERATING METHODS

Intermittent Vs. Continuous Operation.—Two standard methods are employed for controlling automatic stokers; continuously, by modulating the coal feed and air pressure, to meet the load requirement; and intermittently, by turning the motive power on and off, usually with a thermostatic switch.

In a careful comparison of the two methods, in which the same stoker was operated under the two types of control, there was found to be no difference in the average efficiency, whereas at no point in the efficiency curves was there a difference of more than 3 per cent. (Fig. 2.) Even this latter difference was at very light loads where the increased difficulty of testing may easily account for the differential.

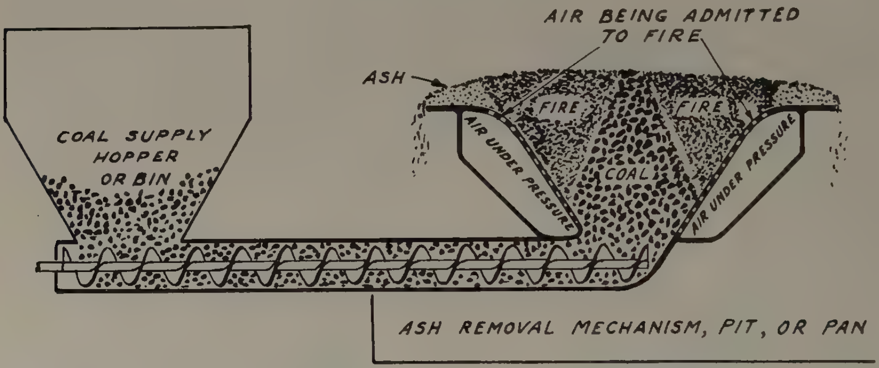
AIR BEING ADMITTED TO FIRE
ASH
COAL SUPPLY HOPPER OR BIN
FIRE
FIRE
AIR UNDER PRESSURE
COAL
AIR UNDER PRESSURE
ASH REMOVAL MECHANISM, PIT, OR PAN

FIG. 1.—PRINCIPLES OF CONSTRUCTION OF LEADING MAKES OF UNDERFEED STOKERS.

An advantage frequently claimed for the intermittent burner, that of less continuous motor operation, is to a large extent, if not completely, offset by the fact that the continuous type can use lighter motors, both because of less frequent starting and lower operating loads.

However, continuous operation may, in some cases, be advantageous in minimizing any tendency towards "cold 70," and as offering a more uniform supply of heat, especially in cases where the heating systems themselves are somewhat sluggish.

Operating Factor.—Estimates, based both upon average weather as compared with extreme conditions as well as upon practical experiences, indicate that an intermittent stoker set to maintain 70° in zero weather will operate approximately 32 per cent of the time. Similarly, a continuous type of stoker will operate at an average point equivalent to 32 per cent of its maximum rating.

Rate of Acceleration.—An ideal rate of acceleration for any fuel-burning system is the rate just equal to the ability of the boiler to absorb the heat generated. Acceleration beyond this point does not serve any useful purpose, and wastes heat to the stack; whereas any deficiency in acceleration results in sluggishness.

The use of thin, rapidly moving fuel beds on underfeed stokers, and the fact that anthracite does not ignite instantaneously, unite to provide an almost ideal condition of pick-up or acceleration.

In actual tests, typical stokerboiler combinations have been brought to a full rate of steaming within 30 min., even after prolonged banking periods; whereas United States Bureau of Standards Specifications (Commercial Standard CS 48–34) require a maximum stack temperature within 60 min. after a 12-hr. period of banking.

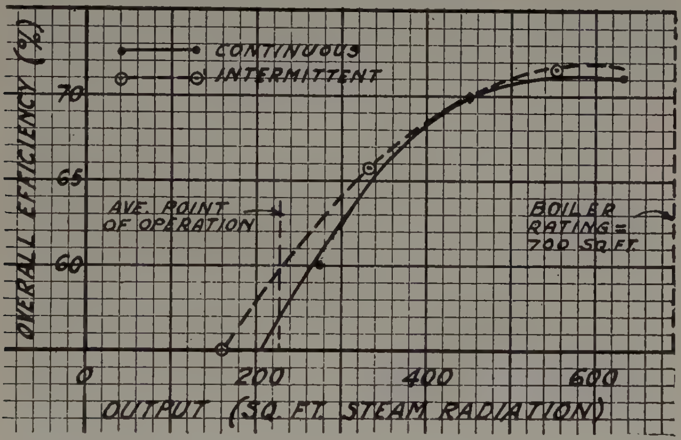

Fig. 2.—Continuous versus intermittent operation.

Advantages of Constant Fuel-bed Thickness.—The constant body of fuel and relatively small air openings of underfeed stokers are distinct advantages from several standpoints: (1) They permit long periods of banking without extinguishment of the fire; (2) they prevent the passage of excessive quantities of air through the boiler during "off periods." The latter is of major importance in preventing any tendency towards a condition of "cold 70," as is the fact that a small fire is always maintained even during banking periods.

Banking.—The exact time of banking without aid from the fan varies widely, with local conditions and design, but is seldom less than 12 hr., with frequently reported instances of 24 hr. A probable fair average is 14 to 15 hours.

Thermal Efficiency

Strictly speaking, the thermal efficiency of a conversion burner should not consider the boiler in which it is installed; it is, therefore, the total heat supplied, minus only the losses to the ashpit and incomplete combustion.

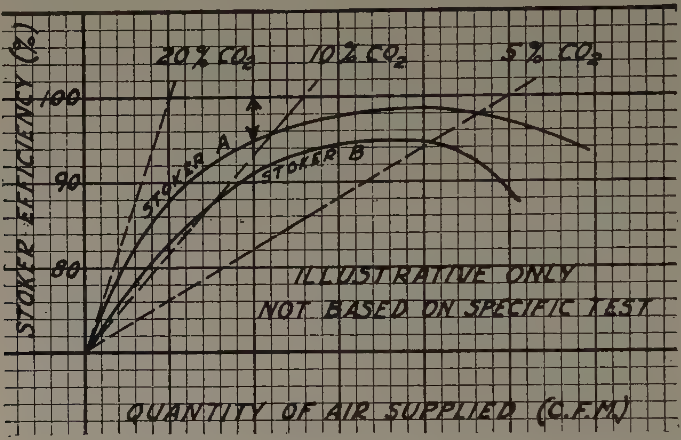

FIG. 3.—RETORT PERFORMANCE CHARACTERISTICS.

Upon this basis, practically all burners show extremely high efficiency figures; as a typical example, 24 tests upon a leading stoker showed an average stoker efficiency of 96.0 per cent.

However, the entire boiler-burner combination must be considered if efficiency figures are to have any practical value.

Fig. 3 shows that as the efficiency of a burner increases, the CO_2 in the resultant gases shows a corresponding decrease. As this increases both the difficulty of recovering the heat, and the unavoidable minimum stack loss, the quality of the products of combustion are indirectly factors affecting stoker efficiency.

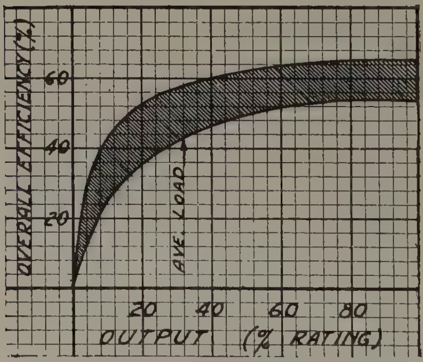

FIG. 4.—EFFICIENCY RANGE (*a*) AND AVERAGE EFFICIENCY (*b*) OF TWELVE BURNERS.

Obviously, the most advantageous practical efficiency exists at the point at which the *total* of the ash and stack losses are at a minimum (Fig. 4). This conclusion has been repeatedly substantiated both in practice and tests, by which it can be conclusively shown that there is a definite balance between stack loss and ash loss. On either side of this balance, a decrease of one loss will inevitably result in an increase in the other. While the exact point of this balance varies slightly with stokers of different tuyere design, a total of the two losses is usually at a minimum, with not less than 10 per cent nor more than 12 per cent CO_2

Attempts to increase the CO_2 beyond this point invariably increase the ash losses, whereas lower percentages of CO_2 waste heat unnecessarily to the stack.

HOUSE-HEATING EFFICIENCY

Since a large part of the losses reported upon the former basis are actually used as a part of the heat input to the house, the stoker boiler efficiency is considerably lower than the actual house-heating efficiency. For example, a large part of the radiation loss is reclaimed in heating the cellar and floor, whereas an inside chimney will dissipate a substantial part of its heat to the adjoining rooms.

Field measurements in five stoker-fired houses indicated an average house-heating efficiency of 85.3 per cent, with a minimum of 81.8 per cent and a maximum of 91.1 per cent.

Unfortunately, actual house-heating efficiencies are so dependent upon local conditions that it is advisable to report test data upon a basis of efficiency at the nozzle of the boiler. (All references to efficiency in this report are upon this basis, except as otherwise noted.)

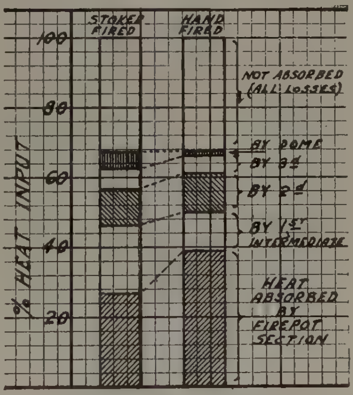

FIG. 5.—HEAT ABSORBED BY VARIOUS BOILER SECTIONS.

As house-heating efficiencies are the final factor in determining fuel consumption, this is an extremely important point, since *savings and consumption data cannot be based upon a direct comparison of boiler efficiencies.*

EFFECT OF BOILER DESIGN

The design of the boiler is important in determining the over-all efficiency of the system. Because of the substitution of a self-contained fire for the direct contact of a hand-fired furnace, stoker installations show a greatly decreased furnace absorption. Thus, if hand-fired loads and efficiencies are to be equaled, intermediate sections are more essential than with hand firing.

To illustrate this point, a vertical cast-iron boiler was so connected that the heat generated in each section could be measured separately. (The entire boiler included one firepot unit, one crown-sheet section, two intermediates and a dome.)

Fig. 5 and Table 1 show that the furnace side-wall absorption of the stoker-fired unit was only 26.4 per cent of the heat input as compared with 39.9 per cent under the same conditions but with hand firing. With the particular boiler tested this difference was largely overcome by a 7.5 per cent increase in the crown-sheet absorption with the stoker unit, so that by the time the gases reached the stack almost identical amounts of heat had been absorbed by the hand-fired and stoker systems. However, in other boilers tested less efficient crown sheets allowed this entire difference of side-wall absorption to remain as a permanent loss.

TABLE 1.—*Heat Absorbed by Various Boiler Sections*

Absorbed by	Stoker-fired	Hand-fired
Furnace walls	26.4	39.9
1st or crown sheet	19.3	11.8
2nd section	10.9	8.5
3d section	6.6	4.9
Dome	4.1	2.3
Total percentage of input absorbed	67.3	67.4

The design solution for this problem is, of course, the use of boilers such as those commonly designed for gas and oil, in which major dependence is placed upon absorption within the flue passages rather than in the furnace. The advantages of one such boiler, as compared with a common vertical hand-fired cast-iron boiler, are shown graphically in Fig. 6.

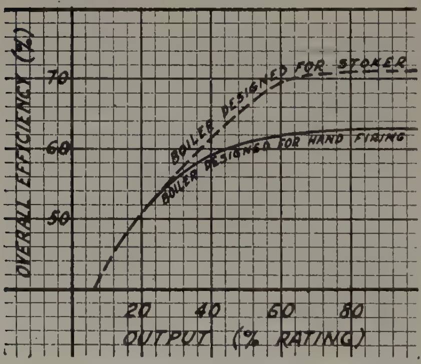

FIG. 6.—EFFECT OF BOILER DESIGN ON EFFICIENCY.

USE OF BAFFLES TO INCREASE ABSORPTION

In the field where boilers obviously cannot be redesigned, refractory baffles are commonly used to overcome this loss, and tests have shown them to range in effectiveness from approximately 8 to 10 per cent increased absorption in the case of boilers having no intermediate sections, to practically zero in boilers amply supplied with secondary absorbing surface. In the latter case the baffles tend to change the distribution of absorbed heat back to conditions more nearly approximating those of hand firing but without final benefit.

Effect of Adjustment.—An extensive series of tests indicates that the difference between the best possible adjustment, as obtained with instruments and painstaking trial and error; and the average field adjustment dictated by experience will be approximately 10 per cent of both the efficiency and output of automatic stokers.

TABLE 2.—*Comparative Data and Characteristics of Domestic Stokers*[a]

No. 1 BUCKWHEAT USED IN ALL TESTS

LINE			TUYERE DIAMETER	BOILER USED IN TEST	AT MAXIMUM COAL FEED								AT MAXIMUM EFFICIENCY								AT MINIMUM COMBUSTIBLE IN ASH								LINE
					EFFICIENCY	STEAM RADIATION	COMBUSTIBLE IN ASH	COAL PER HOUR	COAL PER SQ. FT.	RADIATION PER LB. COAL	ELECTRICITY PER NET TON	CARBON DIOXIDE	EFFICIENCY	STEAM RADIATION	COMBUSTIBLE IN ASH	COAL PER HOUR	COAL PER SQ. FT.	RADIATION PER LB. COAL	ELECTRICITY PER NET TON	CARBON DIOXIDE	EFFICIENCY	STEAM RADIATION	COMBUSTIBLE IN ASH	COAL PER HOUR	COAL PER SQ. FT.	RADIATION PER LB. COAL	ELECTRICITY PER NET TON	CARBON DIOXIDE	
1			18	7	56.5	1135	40.0	31.0	17.6	31.7	9.5		60.0	750	29.2	22.5	12.8	33.8	14.5	7.9	56.0	500	20.0	16.0	9.1	31.4		6.8	1
2			15	10	34.6	485	61.0	28.0	22.8	19.3			34.5	485	61.0	28.0	22.8	19.4		6.0									2
3			18½	8	56.7	590	42.0	18.8	10.0	30.0	12.9			NOT ADJUSTABLE								NOT ADJUSTABLE							3
4			15½	10	53.0	625	40.0	24.8	19.5	29.8	12.0		56.2	460	37.0	17.0	13.8	31.6		9.0	54.0	300	36.0	11.0	8.9	30.6		7.3	4
5			19	10	51.8	654	26.2	24.0	12.3	29.2			54.5	500	22.0	16.7	8.5	30.6		8.2	54.5	500	22.0	16.7	8.5	30.3		8.2	5
6			16	10	60.2	775	43.0	24.0	17.1	33.9	7.9		62.5	600	43.5	18.4	12.8	35.2	9.6	11.2	61.5	500	41.0	15.8	11.3	34.5		10.0	6
7			9	OWN	79.0	280	43.6	6.5	15.5	44.4	17.6		79.0	280	43.6	6.5	15.5	44.5	17.6	13.0	69.8	160	41.1	2.8	6.7	39.2		6.8	7
8			13	1	48.5	423	62.6	16.3	13.9	26.2	26.0			NOT ADJUSTABLE								NOT ADJUSTABLE							8
9			15	10	60.5	685	40.0	20.8	16.8	34.0	26.4		60.5	685	40.0	20.8	16.9	34.0	26.4	10.5	50.0	235	33.3	7.1	5.8	28.2		5.7	9
10			18	7	49.9	1000	52.2	37.4	21.2	28.0	15.7		65.5	515	39.4	14.7	8.3	36.8	24.8	9.8									10
11			17	8	56.0	1090	57.6	37.7	24.2	31.4	9.8		64.0	800	17.0	22.0	14.0	36.0	19.0	7.8	61.2	550	12.0	16.0	10.2	34.4		5.5	11
12			15	1	51.0	616	24.0	23.0	18.7	28.6	12.0		53.0	450	24.0	17.0	13.8	29.8	18.0	7.0	51.0	300	20.0	12.0	9.7	28.6		6.0	12
13			17	7	55.8	613	37.3	20.3	12.9	31.4	12.0		61.0	470	31.0	15.3	9.7	34.2	14.0	9.4	55.7	445	26.8	14.7	9.3	31.4		6.9	13
14			14	7	56.7	658	53.5	21.2	19.8	31.9	9.6		59.4	611	45.9	18.5	17.3	33.4	11.1	10.4	57.5	400	36.5	12.0	11.2	32.4		8.3	14
15			19½	10	59.0	1140	48.0	32.2	15.5	33.1	12.0		63.1	600	36.5	16.6	8.0	35.5	16.0	7.5	63.0	500	32.0	16.2	10.3	35.4		8.9	15
16			13	1	52.5	501	37.9	17.0	18.5	29.5	13.9	15.5	56.5	415	44.6	13.1	14.3	31.6	16.3	12.1	51.8	226	28.1	7.78	8.46	29.2	30.8	6.6	16

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[a] Includes all stokers, irrespective of approval; as tested in the Anthracite Institute Laboratory.

Effect of Coal Size upon Stoker Performance

As practically all domestic burners are sold for either rice or buckwheat, a study of the relative performance with the two sizes is of special interest.

Tests were conducted upon *commercial* sizes of buckwheat, rice and barley. Each size was screened to eliminate excessive amounts of both undersize and oversize, but was allowed to retain standard percentages of both. (Standard Anthracite Institute sizes are shown in Table 3.)

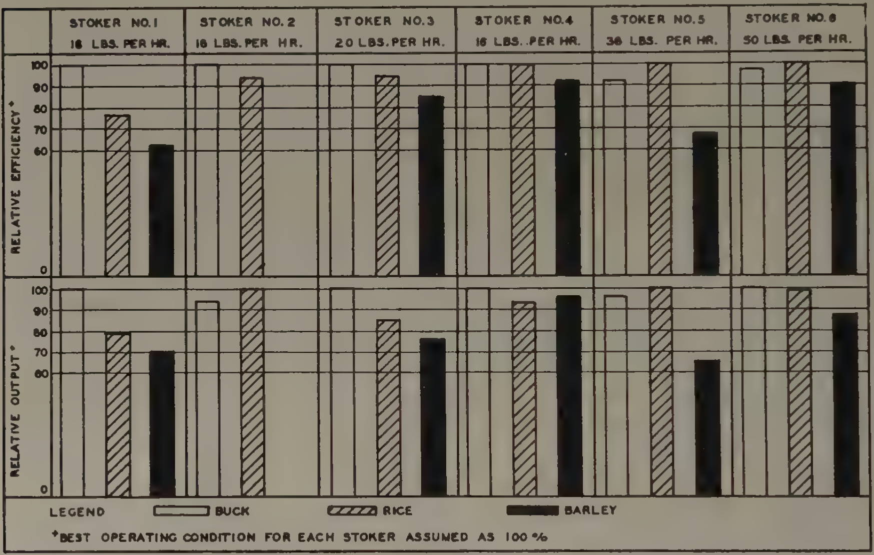

Fig. 7.—Performance characteristics of various approved stokers with buck, rice and barley anthracite.

Each size was burned (with its correspondingly correct air adjustment) upon each of six models of two leading manufacturers. The results of these tests as averaged for each size are given in Table 4, which indicates that the average efficiency of the six stokers tested was some 3 per cent less with rice than with No. 1 buckwheat. However, a correct adjustment of air with rice is considerably more difficult than with No. 1 buckwheat, and, since the table refers to absolutely correct air adjustments, a somewhat greater difference would usually be found in actual practice.

Furthermore: *Since the output with rice is less than with buckwheat, the use of rice should be avoided with overloaded heating systems.*

Barley usually is not suitable for domestic stokers because of its naturally decreased efficiency and output, extreme difficulty of air adjustment and tendency to clinker or burn in spots in underfed retorts.

TABLE 3.—*Standard Anthracite Institute Specifications for Sizing*

	Test Mesh, In.	Oversize Maximum, Per Cent	Undersize, Per Cent	
			Max.	Min.
Buckwheat	Thru 9/16 Over 5/16	10	15	7½
Rice	Thru 5/16 Over 3/16	10	15	7½
Barley	Thru 3/16 Over 3/32	10	20	10

TABLE 4.—*Relative Stoker Performance with Various Commercial Coal Sizes*[a]

Coal Size	Buckwheat	Rice	Barley
Relative efficiency, per cent	100	97	67
Relative output, per cent	100	94	67

[a] Each figure is an average result for six stokers. Individual results of each stoker are shown in Fig. 7.

Special Stoker Rice Coal.—The results previously discussed refer to coal that was prepared in accordance with Anthracite Institute standards, which allow 10 per cent oversize and 15 per cent undersize (in buckwheat and rice). A second series of tests used a specially prepared stoker rice from which practically all oversize and undersize had been removed. A Model M Electric Furnace-Man was used and, although the tests were not extended to include other stokers, it is felt that they are typical and indicative of the expectancy with other models. Table 5 compares these results (commercial buck and special stoker rice) and indicates that there is no appreciable difference between the two sizes when the rice has been carefully prepared to exact screen dimensions.

TABLE 5.—*Relative Stoker Performance with Commercial Buck and Specially Prepared Rice*[a]

Size	Commercial Buckwheat	Stoker Rice
Efficiency, per cent	57.6	57.3
Steam radiation, sq. ft.	485	477
Carbon dioxide	7.97	7.56
Combustible in ash	30.1	34.5

[a] Rice was specially prepared by removing *all* undersize and oversize. Each column represents an average of six tests at six rates of coal feed.

Accidental Variations in Size.—All previously described tests were conducted with the air accurately and correctly adjusted for each individual test. It is, however, of further interest to observe the effect of accidental variations in coal size upon stoker performance; in other words, to determine performance upon a size or mixture of sizes which has been changed since the stoker was last adjusted. It was found that in a typical stoker, which had been correctly adjusted to burn buckwheat of an exact size, both the efficiency and output dropped approximately 1 per cent for every 10 per cent of *either* oversize or undersize added. The combustible in the ash increased approximately 3 per cent for every 10 per cent of *either* oversize or undersize. The effect of oversize and undersize upon a stoker adjusted for rice coal was similar.

Conclusions.—It is concluded that the accuracy of sizing is of great importance in securing proper stoker operation. *Any* change in size should be followed by corrections to the air adjustment of the stoker. (These should be made by a competent service engineer.)

The use of rice should be recommended only when the heating system is of unquestioned ample capacity. As adjustments are somewhat more critical with rice than with buckwheat, rice should be used only where competent service attention is available.

VARIATIONS IN RATE OF COAL FEED

Frequently there is a considerable variation in the rate of coal feed with different kinds and sizes of coal. While sometimes this may be due to changes in size and changes in uniformity of size it is more likely to result from differences in the specific gravity of various kinds of anthracite. As shown in Table 6, two coals screened to exactly the same size were fed with a 12.7 per cent difference in weight, with the same rate of stoker feed.

TABLE 6.—*Typical Variation in Rate of Feed*

	Coal A	Coal B	Per Cent Difference
Lb. fed per hr.	59.8	48.8	22.5
Specific weight (lb./cu. ft.)	51.2	45.4	12.7

This factor is important in rating stokers, for since heat value is not proportionate to specific gravity, all stokers should be rated upon a basis of the lightest (or most bulky coal) which they are to burn.

EFFECT OF SETTING HEIGHT

There are marked differences of opinion, concerning the effect of variation in furnace volume upon stoker performance. Therefore tests were conducted to secure relative data with a wide range of setting heights. Unfortunately, the only boiler available for these tests was considerably larger than the capacity of the stoker, thus resulting in abnormal efficiencies due to low stack temperatures. It is believed, however, that the relative picture secured was not affected by this condition and that it can be used, therefore, as a rough guide to expectancy.

The sketch shown as Fig. 8 illustrates the relative position of the crown sheet, water leg and tuyere pot.

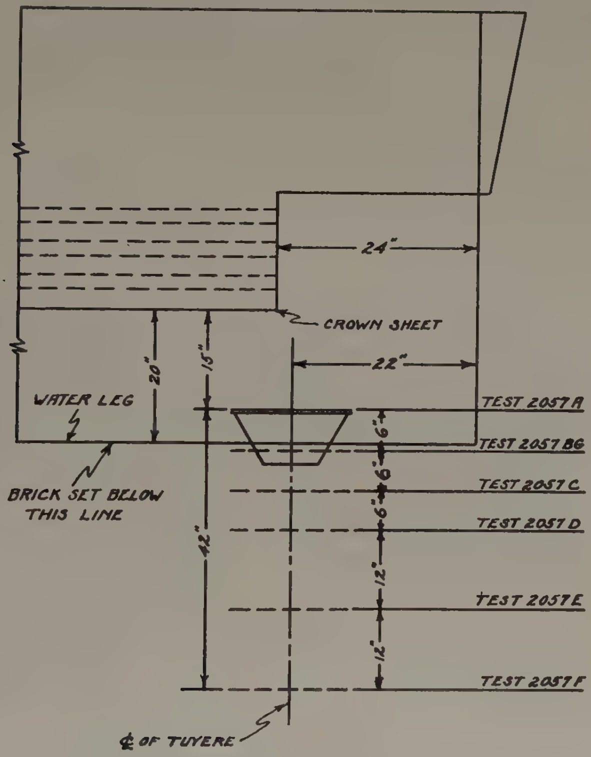

FIG. 8.—COMPARISON OF SETTING HEIGHTS IN STOKER INSTALLATIONS.

A series of six running-start tests, each of 8 hr. duration, was conducted. The only change made between the tests was that of stoker location and a thorough cleaning of the boiler.

Standard measurements were secured from which both direct evaporative and heat-balance computation of efficiencies could be secured. In addition, the fly-ash and refuse in the tubes were carefully removed by a vacuum cleaner between each test and measured.

Fig. 9 presents a detailed picture of the results of these tests. The efficiency of the boiler-stoker combination was at a maximum when the distance from the crown sheet to the top of the tuyere approximated 29 in. The efficiency, however, was uniformly high with all settings less than 35 in.

As the water leg of the boiler extended for only 20 in. below the crown sheet, decreases in both efficiency and output undoubtedly were due to greater increased radiation loss through greater areas of unjacketed brickwork rather than to any factor of furnace volume. A study of the heat balances indicates that this is correct, since the ashpit loss actually decreased as the setting height increased, the stack loss remained almost constant and no carbon monoxide was noted in any test. On the other hand, the radiation loss increased from a minimum of 9.0 to 15.1, which corresponded almost exactly with a decreased efficiency of 5.5 per cent.

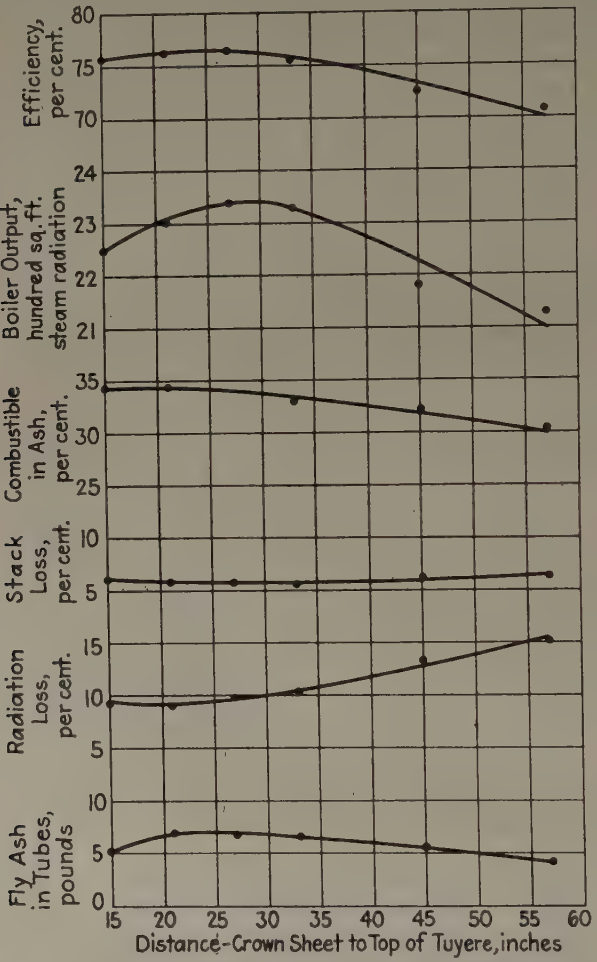

FIG. 9.—EFFECT OF SETTING HEIGHT IN STOKER-FIRED BOILER.

Relative Cleanliness with Various Setting Heights.—As shown on the curve sheet, the setting height has surprisingly little effect upon the small amount of fine coal blown from the tuyere into the tubes. Even the small difference shown probably is due to the fact that with increased setting heights a greater percentage of this material fell back on the tuyere or into the ashpit.

The refuse removed from the tubes was partially burned coal analyzing from 60 to 75 per cent combustible. No traces of soot or other greasy material was noted with any anthracite fire.

It is concluded, therefore, that the only factor of setting height that affects either the efficiency or the output is that, after the plane of active fire passes below the bottom of the water leg, the resultant increased uncooled surfacer increase the radiation loss.

Rate of Combustion

The most important single factor of stoker design is a determination of the amount of fuel to be fed per square foot of tuyere area. In a recent comprehensive paper, Mr. Percy Nicholls proved conclusively (p. 183, this volume) that a definite *maximum* rate of combustion existed for all solid fuels. A careful analysis of all available stoker results substantiates this and coordinates it with current practice by showing that in no case did the *burning* rate of any stoker exceed 20 lb. per square foot per hour (Fig. 10). (In this connection, it is extremely important not to confuse pounds of coal *fed* with pounds of coal burned, since in many cases considerably more coal was fed only to be discharged as unburned combustible in the ash.)

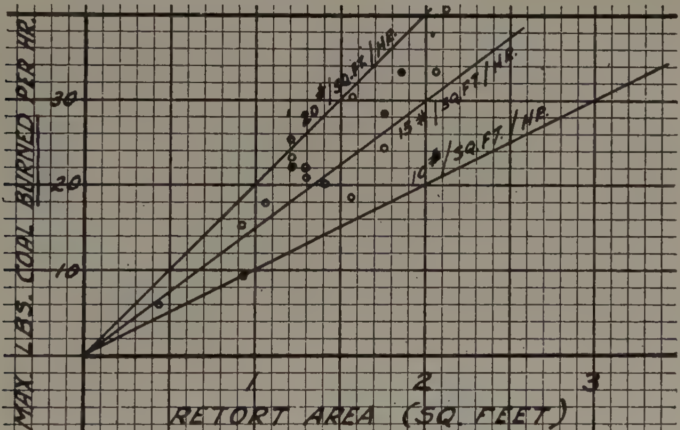

Fig. 10.—Relation between retort size and burning rate. (Curves are calculated rates and are not plotted from test points.)

Consumption of Electricity

The electrical consumption of domestic burners as expressed in terms of kilowatts per ton of fuel fired varies widely in accordance with the size and type of the motor and rate of coal feed (Fig. 11). Consumptions as low as 10 kw. (at full rating) have been noted, whereas 50 to 60 kw. is not uncommon on relatively large stokers operated at their lowest rate of coal feed. An average of from 20 to 30 kw-hr. is sufficiently correct for seasonal estimates of cost (which may be based upon an operating factor of 32 per cent).

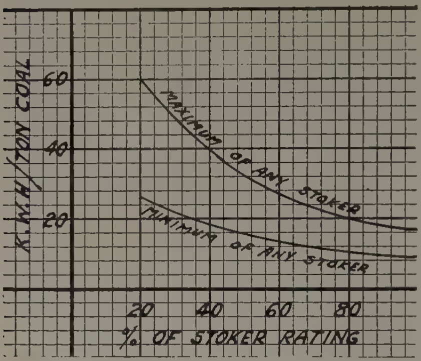

Fig. 11.—Range of electrical consumption.

Selection of the "Best" Stoker

It is almost certain that no report of stoker performance would be accepted as being complete unless it presented data to show which of the several stokers on the market is the "best."

Entirely aside from any consideration of the wisdom of attempting such a statement, there are so many interrelated factors to be considered in selecting a stoker that it is extremely doubtful whether any one stoker could possibly be singled out to be considered better than the others. This report has shown that a number of factors of performance are of importance in the selection of a stoker; even a casual comparison of two stokers from several standpoints will show that whereas one may be better in some respects these will be balanced in others. In addition, the following factors should be given serious consideration:

1. Probable service attention required.
2. Factory support to dealers.
3. Service facilities available.
4. Care and skill in installing.

A stoker that satisfactorily passes these requirements, and at the same time can comply with the standards of construction, installation and performance as specified by the United States Department of Commerce, can be expected to provide years of satisfactory heat with a maximum of safety, economy and comfort. With such advantages, its relative position among its competitors is a trivial consideration.

DISCUSSION

(*Cadwallader Evans, Jr., presiding*)

Member.—What is the status of automatic heat, using nut coal?

A. J. Johnson.—The D. L. & W. Coal Co. has developed a chain-grate stoker with a cast-iron type of boiler, and has sold a number; a number have been installed on, one might say, a semi-experimental basis; that is, they are being placed in the field, but I do not believe that they are being widely sold commercially.

The Philadelphia & Reading Coal and Iron Co. has developed an interesting stoker in which are used a rotating grate and chestnut coal fed from an overhead bin. A number of those stokers also have been sold. In neither case have the stokers reached a point where they may be considered to be actively on the market. That is, while they are being sold, the public has not accepted them to a point that would permit them to be considered as active competitors of either oil or gas.

Member.—How do the efficiencies compare with buckwheat and rice?

A. J. Johnson.—In general the efficiencies are higher than an underfeed stoker for either rice or buckwheat coal.

W. S. Blauvelt,* Troy, N.Y. (written discussion).—The author defines the thermal efficiency of a "conversion burner" as "the total heat supplied, minus only the losses to the ash pit and incomplete combustion." Should he not subtract also any losses due to excess air in the flue gases? The percentage of excess air when burning any given fuel varies with the design and method of operation of the fuel-burning part of any heating appliance, and at any given temperature of the flue gases leaving a boiler, other things being equal, the heat loss in dry flue gases increases as the excess air increases or the percentage of CO_2 decreases. When anthracite containing 79 per cent total carbon as fired is burned with no CO in the flue gases, the percentage of excess air varies with the CO_2 in the flue gases about as follows:

Per Cent CO_2	Per Cent Excess Air	Per Cent CO_2	Per Cent Excess Air
13	45	9	100
12	55		
11	70	8	115
10	85	7	135

If, as frequently is true, the temperature of the air entering the ash pit is about 60° F. and the temperature of the flue gases leaving the boiler is 400° F., the heat loss due to excess air, the result of imperfect combustion may exceed the sum of the losses with another appliance because of unburned CO and excess air.

The word "efficiency" is overworked and underdefined. On page 166 is the statement, "24 tests upon a leading stoker showed an average stoker efficiency of 96.0 per cent." When burning standard quality buckwheat anthracite—79 to 80 per cent total carbon and 11 to 12 ash as fired—the loss in unburned combustible in the ash refuse containing 35 per cent combustible would exceed 7 per cent of the total heat units in the fuel as fired; some explanation of "an average stoker efficiency of 96.0 per cent" would be welcome.

Of the tests reported in Table 2, by far the most interesting is on line 7 "at maximum efficiency." In this test the "efficiency," presumably over-all thermal efficiency of furnace and boiler, is 79 per cent; the next highest is 65.5 per cent.

To the incomplete data in this table I have appended certain assumptions and calculations, which appear reasonable and certainly not unfavorable to the underfeed stoker, in an attempt to compare operating results with those obtained in a test made at the Pittsburgh Experiment Station of the Bureau of Mines (Table 7).

This comparison indicates that the very high efficiency obtained by Mr. Johnson in this test is due far more to the boiler than to the stoker. If the loss of carbon in the ash refuse had been as low as in the Pittsburgh test, other losses remaining as shown, the over-all thermal efficiency of boiler and furnace would have approximated 83 per cent.

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* Hudson Valley Fuel Corporation.

TABLE 7.—*Comparison of Operating Results on Anthracite*

	Reported in *Tech. Paper* 315, Bureau of Mines, Table 4	Reported by Mr. Johnson, Table 2
	Dunning Boiler 303-S, Trial No. 8	Own Boiler, Maximum Efficiency, Line 7
Per cent ash in fuel as fired	11.59	Assumed 11.59
B.t.u. per lb. in fuel as fired	12,667	Assumed 12,667
Carbonaceous matter in ash refuse, per cent	34.15	Reported 43.6
Ash and refuse, per cent of fuel fired	16.95	Calculated 20.55
Per cent of fuel fired, in unburned combustible in ash refuse by difference	5.36	Calculated 8.96
Temperature of air entering ash pit	84° F.	Assumed 84° F.
Excess air in flue gas, per cent	48	Assumed 46
Flue-gas analysis, per cent: CO_2	12.7	Reported 13.0
O_2	6.9	Assumed 6.4
CO_2	0.4	Assumed 0.0
Mean temperature flue gases leaving boiler	237° F.	Assumed 237° F.

HEAT BALANCE ON FUEL AS FIRED

	B.t.u. per Lb. Fuel	Per Cent of B.t.u. in Fuel Fired	B.t.u. per Lb. Fuel Fired	Per Cent of B.t.u. in Fuel Fired
Heat transferred to water	9,155	72.3	Calculated 10,007	Reported 79.0
Heat carried away by steam in flue gases	291	2.3	Assumed 291	2.3
Heat carried away by dry flue gases	518	4.1	Estimated 481	3.8
Heat lost by not burning CO	227	1.8	0	0.0
Heat lost by not burning carbonaceous matter from ash pit and grate	845	6.7	Calculated 1,411	11.1
Undetermined losses	1,631	12.8	477	3.8
Totals	12,667	100.0	12,667	100.0

MEMBER.—Mr. Blauvelt asked the question why air excess should not be considered as a reduction from the total per cent of fuel furnished. Is that correct?

W. S. BLAUVELT.—The excess oxygen in the chimney gas should be a factor in determining the efficiency of any fuel-burning appliance. If the combustion gases in passing through the boiler are cooled to 400° with a room temperature of 60° there is a chimney loss of 300° in the stack gases; if there is no excess oxygen and no combustible in the stack gas, this loss is at a minimum; if, however, one appliance requires for complete combustion but 10 per cent while another requires 100 per cent excess air, the former is clearly the more efficient.

A. J. Johnson.—The function of the stoker is to burn the coal, and the function of the boiler is to absorb the resultant heat. CO_2 is in reality complete combustion, regardless of its percentage; that is, 2 per cent CO_2 is a complete combustion of the coal. Therefore, theoretically (but not practically) a stoker has performed its function satisfactorily even with only 2 per cent of carbon dioxide in the gases. It is the function of the boiler, then, to abstract the heat from that CO_2-laden gas and transform it into useful work. Therefore, in common engineering practice, the grates, or in the case that we are discussing, the stoker, is charged only with delivering radiant heat, and CO_2 from the coal. If it does that without any CO, and without unburned percentages of carbon in the ash, then theoretically it may be considered to be performing its job efficiently. It is up to the boiler from then on to extract that heat. The stack loss is never a function of stoker performance. The stack loss theoretically should be zero and is chargeable not to the stoker but entirely to the boiler. So that my answer would be that regardless of the percentage of CO_2, the boiler, to be 100 per cent efficient, must extract all of the heat from the gases that are given to it.

C. Evans,* Scranton, Pa.—In other words, you shift the whole thing to the boiler.

A. J. Johnson.—Once the heat is extracted from the coal, yes, and using a waste-heat boiler as an example only gas will pass into such a boiler. The gas might be very low in carbon dioxide content, and yet the waste-heat boiler would be of no practical purpose unless it was able to extract the heat from those gases. The efficiency of such a waste-heat boiler—I speak of waste-heat boilers because they are entirely remote from the process of combustion—would be calculated on its ability to extract the heat from the gases supplied to it even if they are at low CO_2.

W. S. Blauvelt.—If you take Mr. Johnson's definition, then, a stoker that will waste 20 per cent of the carbon in ashes, and has no carbon monoxide with 300 per cent excess air, is equally efficient as a stoker that burns the ash down to no combustible in the ash and with only a fraction of one per cent oxygen in the chimney gases and no CO.

A. J. Johnson.—I said also no loss in the ashes.

W. S. Blauvelt.—You left that out in your remarks, I think, Mr. Johnson.

C. Evans.—Has the Anthracite Institute found it possible to make any endeavor to determine the over-all efficiency of stokers and boilers, or is there such a great variety of boilers that it has not been possible?

A. J. Johnson.—We are making a study of house-heating efficiency now. We have equipped one house with instruments and have installed a gas burner for six weeks, an oil burner for six weeks, and now an anthracite stoker for six weeks. We are checking the heat against weather bureau records, and so forth, in order to determine the house-heating efficiency with those three fuels. However, the house-heating efficiency will depend to a very large extent on the house itself. For instance, an outside chimney would reclaim an entirely different amount of heat from the gases supplied at its base than would an inside chimney, so that there will be a wide range in so far as house-heating efficiencies are concerned. These tests are being made in one house, therefore the only variable will be the weather conditions, which are being very carefully recorded.

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* General Manager, The Hudson Coal Co.

H. J. Rose,* Pittsburgh, Pa.—Before we leave the subject, I should like to revert to the earlier discussion regarding automatic means of burning a larger size, such as chestnut. Two types were mentioned. I should also like to point out that for decades people have been burning the larger size of anthracite in appliances of the magazine-feed type, which do burn the larger sizes with efficiency and comfort with low capital expenditure, no moving parts and no cost for electricity. For average operation one needs to refuel them only once every 24 or 36 hr. When considering automatic combustion of anthracite by means of stokers, we should not overlook magazine burners, the good old standby for burning anthracite.

E. L. Buller,† Scranton, Pa. (written discussion).—From the fuel standpoint, this equipment will operate with the utmost success providing the fuel does not contain an excess amount of oversize or undersize, or to be more explicit, if we may borrow a term from the waterworks profession, the ideal domestic stoker fuel would have a uniformity coefficient of 1.0 or nearly so.

It can be readily seen that when a fuel such as the barley size of anthracite coal, which is made through a 3/16-in. round mesh and over a 3/32-in. round mesh (note the wide range in size), is conveyed through a hopper and then, by means of a horizontal worm or plunger, to a retort where another change of direction must take place, a segregation of particles of unequal size is bound to occur. When this occurs it is impossible to maintain good combustion because the air supply to the stoker can never be properly adjusted to meet the air demands for the size of coal that is being burned. At one period the stoker may be burning the smaller sizes of the segregated material and at another period the stoker may be burning the larger particles of the material, while the air adjustment has been made for the particle of average size. Either clinkers or a high-carbon ash is then the result and a consumer complaint immediately follows.

Recently, tests were conducted in a domestic underfeed stoker burning No. 1 buckwheat and giving 65 per cent over-all efficiency with an actual ash percentage of 18.2 per cent. Then the mixtures of No. 1 buckwheat and rice coal shown in the accompanying table were fed to the same stoker. For brevity, the total heat balance is not shown here, but the results obtained were by no means on a par with those obtained when a straight fire (No. 1 buckwheat) was used, because of the segregation of the sizes that takes place, making it practically impossible to arrive at a definite air setting that will satisfy both sizes of fuel.

Mixtures of Buckwheat and Rice

Per Cent No. 1 Buckwheat Through 9/16 In.; Over 5/16 In.	Per Cent Rice Through 5/16 In.; Over 3/16 In.
80	20
70	30
60	40
50	50
40	60
30	70

From an ash standpoint, the constituents of the ash that govern the flue temperature are of the utmost importance. It is much better to have a fuel with from 10 to 12 per cent ash, which is powdery and easy to convey, than to have a fuel with 8 per cent ash when this low-ash fuel has a very low fusion point. Anthracite coal for use in domestic stokers must be given the utmost attention in preparation as to both ash and sizing.

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* Senior Industrial Fellow, Mellon Institute of Industrial Research.

† Research Engineer, The Hudson Coal Co.

INSTALLATION

When considering the setting heights, it is advantageous (as is indicated in Figs. 12 and 13 and the data that follow) to locate the retort of the stoker in the same plane as the bottom of the water legs of the boiler.

Because engineers do not agree as to the relative merits and amount of pitting a domestic stoker should be subjected to, the following study was made for the benefit of our service men, and dealers who are interested in the sale and naturally the proper installation of this equipment.

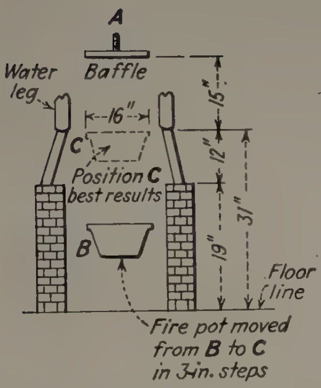
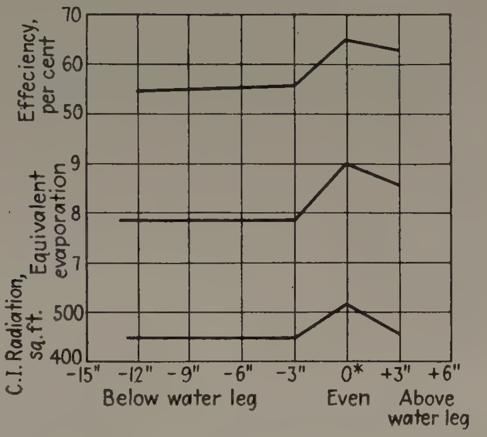


FIG. 12.—SETTING OF BOILER AND STOKER.



* Retort even with bottom of water leg of boiler

FIG. 13.—RECORD OF LOAD AND EFFICIENCY.

The stoker and boiler used in this investigation were of a standard make, both having been on the market for several years. Since neither the stoker nor boiler seriously affects the results to be obtained, the names are not considered essential.

The boiler was mounted on a brick setting that raised the water leg of the boiler 31 in. from the floor, as shown in Fig. 12. The stoker was installed with the firepot 12 in. below the water leg, with a circular baffle suspended 15 in. above the burner pot.

TABLE 8.—*Load-carrying Ability and Efficiency*

	Below Water Leg			Above Water Leg	
Position of stoker retort	−9	−6	−3	0[a]	+3
Fuel, lb. per hour	14.6	14.3	14.1	13.90	13.40
Equivalent evaporation, lb. water per lb. coal	7.89	7.97	7.92	9.09	8.64
Cast iron radiation generated, sq. ft.	466.0	460.0	454.0	512.0	470.0
Rating developed, per cent	66.5	65.6	64.9	73.0	68.0
Efficiency B. F. stoker	56.0	57.3	57.0	65.0	62.8
Carbon ash, per cent	11.0	9.9	12.1	10.0	11.0
Ash, per cent	89.0	90.1	87.9	90.0	89.0

[a] Retort even with bottom of water leg of boiler.

The stoker was then operated for 12 hr. at this location, feeding in each case approximately 14.5 lb. per hour.

After each 12-hr. run, the burner and baffle were raised in 3-in. stages until the top of the retort was raised 3 in. above the water leg. As indicated in Fig. 13, the developed load and the efficiency increased to a maximum when the top of the stoker firepot was placed on a level with the bottom of the water leg of the boiler.

It is appreciated that all installations cannot be made so that the top of the retort is in the same plane with the bottom of the water leg of the boiler, because often the ash pit is not deep enough for such an installation. In that event, if it is at all possible, the floor should be taken up so that the stoker may be dropped to the proper level.

As indicated by Fig. 13 and the data of Table 8, load-carrying ability and efficiency are also lost as the firepot of the stoker is raised into the boiler firepot.

Principles of Fuel Beds*

By P. Nicholls†

(Pennsylvania, State College Meeting, October, 1934)

Though the burning of fuels extends far back into antiquity, and though fuel beds are the most common and widely distributed example of chemical actions and engineering practice, there has been little attempt to group fuel beds by types or to define the principles associated with them.

This paper does not profess to cover completely the subject of fuel beds. It suggests some of the over-all principles of the two more common types. No new investigational data are included, but the deductions are largely based on a recent publication of the U.S. Bureau of Mines[1].

Definition of Types

There are three primary types of fuel beds: (1) Underfeed, in which the fuel and the air move in the same direction; (2) overfeed, in which the fuel and the air move in opposite directions; (3) crossfeed, in which the fuel and the air move at right angles to each other.

The principle that fixes the type is the relation between directions of the flow of fuel and of air. Fig. 1 shows the three types diagrammatically. All other types are compounded of these three in a simple or more or less complex manner.

It does not follow that these three type designations are limited to the methods used in representing them in Fig. 1; they may be turned in any direction, and as long as the relative directions of the fuel and air are the same the type is not changed. For example, *A* as shown resembles a pot-type stoker, but the air and fuel can move downward as in the so-called smokeless heating boiler furnace. Also, *C* can be placed horizontal, as in chain-grate stokers. For this reason, the terms used to designate the three types are not all that could be desired; more logical ones could be coined, but those used probably are satisfactory until the discussion of fuel-bed types becomes more common.

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* Published by permission of the Director, U.S. Bureau of Mines. Manuscript received at the office of the Institute Oct. 25, 1934.

† Supervising Fuel Engineer, Pittsburgh Experiment Station, U.S. Bureau of Mines.

[1] P. Nicholls: Underfeed Combustion, Effect of Preheat, and Distribution of Ash in Fuel Beds, U.S. Bur. Mines *Bull.* 378 (1934).

The movement and disposal of the ash do not affect the type; the released ash will flow in the same direction as the fuel unless it becomes fluid, when gravity and the temperature of the zones into which it flows will influence its motion.

The planes of ignition of the three types are shown at the junctions of the fresh fuel with the burning bed. However, the old question comes up as to what is meant by "ignition." The more common meaning of "starting to burn," that is, to combine with oxygen, is not satisfactory. For instance, a piece of coke placed on top of a deep hand-fired bed may get red hot and yet not be able to combine with oxygen because that has been used up by the fuel below, or even to react with CO_2, because all of that has been converted to CO. It is more satisfactory, in connection with fuel beds, to consider ignition as the attainment of a temperature at which the fuel would burn rapidly if oxygen were available; otherwise, the plane of ignition is indefinite. The exact temperature used is not of much practical importance, because when burning starts the rate of increase in temperature is very rapid and the change in the position of the plane of ignition is very small. In practice, the term "rate of burning" usually refers to the pounds of fuel burned. In this paper it means the pounds of combustible gassified; so also "rate of ignition" means the the pounds of available combustible brought to ignition temperature.

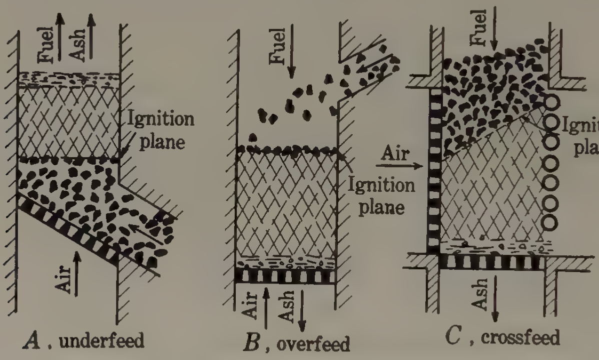

FIG. 1.—THE THREE PRIMARY TYPES OF FUEL BEDS.

The terms "unrestricted ignition" and "restricted ignition" are used and are important. Unrestricted ignition means that no conditions that limit the rate of ignition are imposed, that this rate is free to be whatever the type of fuel permits. Thus, in *A* of Fig. 1 there is unrestricted ignition. Restricted ignition means that conditions are imposed that limit the rate of ignition. Thus in *B* the rate of ignition cannot be faster than the rate of feeding of the fuel; in *C* it may be restricted by the design cutting off the air from the incoming fuel.

An "agitated bed" means one in which the fuel is continuously shaken to avoid the formation of large hollow spaces or to prevent the pieces of fuel from caking together. The term "equilibrium fuel bed" is self-explanatory. It means a fuel bed that, for a constant rate of primary air, maintains a steady character of combustion and thickness.

Underfeed Fuel Beds

In Fig. 1 the underfeed was placed before the overfeed type, although the latter is older and more common, because the underfeed bed is better adapted than the overfeed to being operated with the minimum number of restrictions and to illustrate principles diagrammatically.

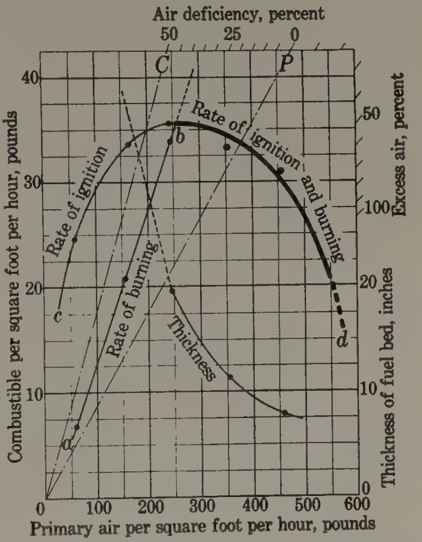

Fig. 2.—Underfeed burning with unrestricted ignition. High-temperature coke, 1 to 1½-in. size; 80° F. air temperature.

Underfeed burning is commonly associated with underfeed stokers, but as usually operated these stokers do not have true underfeed beds and their burning may be a combination of the three primary types of bed. The author has used the term "pure underfeed burning" to distinguish a bed as represented by *A* from underfeed stokers. In *A* no restriction is imposed on the burning resulting from a given rate of air supply other than the effect of the walls—which can be made small by making the area of the bed large—and the loss of heat by radiation from the top surface of the bed. If the ignition plane reaches the grates, there is restricted ignition. Small, pot-type stokers may approach pure underfeed burning under certain conditions, but usually the rate of air supply is too low. The Hawley downdraft bed (Fig. 4) is the one type having pure underfeed burning.

We will therefore consider a type of underfeed bed as shown by *A* of Fig. 1, in which all the primary air—that is, air supplied below the bed—passes through the plane of ignition. Fig. 2 is a convenient type of plot to illustrate the principles. The horizontal axis is the pounds of primary air per square foot per hour. The vertical axis is pounds of combustible per square foot per hour and may be used to represent combustible fired, combustible ignited, or combustible gasified.

Line *OC* fixes the theoretical maximum rate of combustion for each rate of air supply. When a hydrogen-free fuel is burned, it is the line which represents only CO and N_2 in the products of combustion. When fuels with volatile matter are burned, all the free hydrogen in the fuel could be present as H_2, but the line is not quite as definite as with high-temperature coke because it might be assumed that there is CH_4, and even carbon as soot in the gases. All points of operation on line *OP* have perfect combustion—that is, only CO_2, H_2O and N_2 in the gases; points of operation left of line *OP* require secondary air, and those right of it have excess air. The scale on the top permits the secondary requirements at any rate of burning to be read off. The exact position of lines *OC* and *OP* is not the same for all fuels.

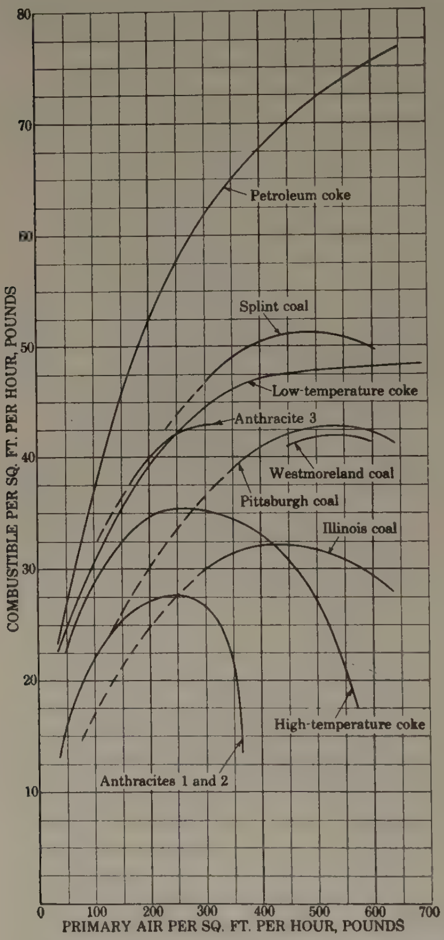

FIG. 3.—RATES OF IGNITION OF FUELS COMPARED; 1 TO 1½-IN. SIZE, 80° F. AIR TEMPERATURE.

The plots on Fig. 2 are from tests of a high-temperature coke of 1 to 1½-in. size, burning with unrestricted ignition. Curve *cbd* is the rate of ignition and *abd* the rate of burning. At 50 lb. air rate the coke ignited faster than it burned, therefore the fuel bed was continually increasing in thickness. As the air rate increased the difference between the rates of ignition and burning decreased, until at 260 lb. of air they were the same. For higher air rates, the rates of burning could have followed along the dotted extension of line *ab*, but because they cannot be greater than the rate of ignition, they are fixed by curve *bd*.

For air rates less than that of the intersection of the curves at point *b* there is not equilibrium burning, because the thickness of the bed is increasing; for air rates greater than that of *b* there is equilibrium burning, and for a given air rate the thickness of the live bed is constant. Fig. 2 includes a curve for the thickness of the bed under the conditions of test; the thickness decreases with increasing air rate.

Fig. 2 shows that with this fuel the maximum rate of burning was with 270 lb. of air; that increasing the air rate above this decreased the rate of burning; and that at about 570 lb. of air the burning could not be maintained and the fire went out. Increasing the rate of feed so that it is greater than the rate of ignition would not affect the rate of burning; what would happen is that the plane of ignition and the burning fuel above it would rise correspondingly higher above the grates.

The burning of all fuels on the underfeed principle and with unrestricted ignition can be represented in the same manner as Fig. 2, but the rate-of-ignition curves differ materially for different fuels and for different sizes of the same fuel. Fig. 3 shows the ignition curves for nine fuels, all of 1 to 1½ in. in size. All the curves reached a maximum and then started to fall off, except for the low-temperature and petroleum cokes.

It is not possible to maintain beds with uniform burning at low air rates with coals that cake, because the caking causes holes to form in the bed; however, the principles are the same as those for free-burning coals, and if the pieces could be kept separate the ignition curves would be similar to those of noncaking fuels. In Fig. 3, the dotted parts of the curves for the Illinois, Pittsburgh, and splint coals are not experimental but have been drawn for what they might be with agitated beds.

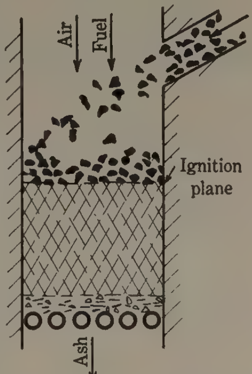

FIG. 4.—UNDERFEED BED WITH RETARDED IGNITION.

Pure Underfeed Burning with Restricted Ignition

To have restricted ignition with a furnace of the type shown in *A* (Fig. 1) it would be necessary for the plane of ignition to be at the grate level, but under such conditions it would not be possible to feed the fuel evenly over the area of the bed. Therefore it is better to picture a type of furnace as shown in Fig. 4, in which the fuel can be fed at any rate desired.

If with any given air rate the fuel is so fed that the temperature of the top exposed layer is never higher than the air temperature, there will be unrestricted ignition, and plots like Fig. 3 would result. The fact that the the surface of the fuel would be supported by water or steam pipes would increase the thickness of the bed a little because of abstraction of heat from the fuel, but it would not influence the rate of ignition unless the bed were thin.

If, with a given air rate, the rate of feed were so reduced that the top layer of fuel in Fig. 4 was at a higher temperature than the air, and if conditions were kept constant long enough, the bed would assume a constant thickness and there would be equilibrium burning with restricted ignition.

Fig. 5 shows the principles involved in restricted ignition. It is for the same fuel as in Fig. 3, and the rate-of-ignition and rate-of-burning curves of that figure, *cbd* and *abd*, for unrestricted ignition are repeated. It has been shown that the burning curve *abd* fixes the maximum rates of operation and that these cannot be exceeded, but by restricting the ignition—that is, the rate of feed—one could operate continuously at any lower rates represented by points within the area *abd*.

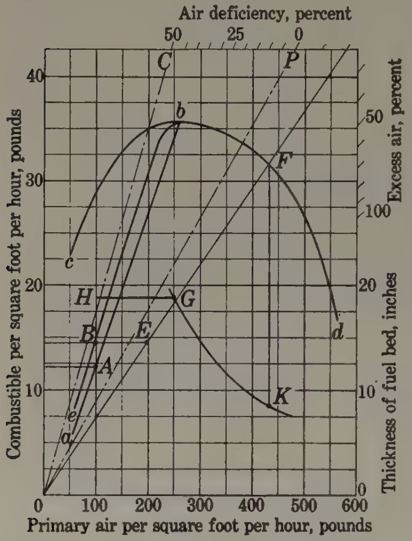

FIG. 5.—UNDERFEED BURNING WITH RESTRICTED IGNITION. HIGH-TEMPERATURE COKE, 1 TO 1½-IN. SIZE; 80° F. AIR TEMPERATURE.

With 100 lb. air rate and unrestricted ignition, one would be operating at point *A* and would burn 12.25 lb. of combustible per hour; but the corresponding rate of ignition would be 29 lb., and there would not be equilibrium burning. If, now, the rate of feed is reduced to 12.25 lb., the rates of ignition and of burning would be the same. However, by reducing the rate of ignition we have reduced the amount of fuel that is heated to the ignition temperature; therefore the temperature through the bed will be higher and the proportion of CO to CO_2 greater. If the rate of feed were gradually increased until the temperature of the top layer of the fresh fuel was just starting to rise, we could operate at point *B* and increase the rate of burning to 14.50 lb. per hour. Note that point *B* cannot be left of the line *OC*. The foregoing can be generalized by saying that curve *ebd* fixes the limits of operation with restricted ignition.

Operating at point *B* would give a burning with high CO and about 40 per cent deficiency of air. This deficiency can be reduced by keeping the rate of feed the same and increasing the air rate; that is, by moving along *BE*. By operating at point *E*, there would be 20 per cent excess air, so that it would not be necessary to supply air above the fuel bed.

The thicknesses of the fuel beds with restricted ignition can be predicted approximately from the principle[2] that the thickness will be the same for all points having the same fuel to air ratios; for example, for points along any line *OF*. Thus, as the thickness for point *F* is known from test, the thickness for point of operation *E* will be the same, or

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[2] H. Kreisinger, F. K. Ovitz and C. E. Augustine: U.S. Bur. Mines *Tech. Paper* 137 (1916) 51.

8½ in., and for point *B* it will be the same as for *G*, or 18½ in. In moving from *B* to *E*, the thickness will move along line *GK*.

The correctness of this method of predicting the thickness will depend on the conditions imposed by the furnace. If the burning is influenced much by the heat absorbed by the side walls, the thicknesses for equilibrium burning corresponding to points of operation on the line *OF* will increase somewhat with decreasing air rate from *F* to *O*, because the quantity of heat released decreases with decrease in air rate but that absorbed by the walls is approximately constant, and thus the proportion absorbed by the wall will be less; thus the temperatures of the fuel bed will be lower and a greater thickness will be required to give the equilibrium rate of burning. The heat used in igniting the fuel will not influence the thickness materially, because with restricted ignition the weight ignited is always equal to that burned.

Overfeed Burning

In overfeed burning, unrestricted ignition—in the sense that the fuel is free to ignite as fast as it can—is not a practical possibility; such a fuel bed would have to be of unlimited height, and all the heat released by the burning of the live bed would be used in igniting the fresh fuel at a rate many times greater than the rate of burning.

Therefore, all fuel beds burning on the overfeed principle have restricted ignition and are controlled by the rate of feed of the fuel; they will assume constant thickness and burn in equilibrium if the rate of feed is not greater than the maximum rate of burning possible with the rate of air supply.

Fig. 6 illustrates the effect of the rate of feed; for simplicity high-temperature coke is again used. The air rate is assumed to be constant and each bed burning in equilibrium.

The upper part of Fig. 6 shows cross-sections of beds of various thicknesses; the lower part is the plot of the total combustible carried, in the gases per pound of primary air plotted against height. In the sections of the beds the top layer is shown as unignited, and a distance marked *T* is that required for the fuel to reach the ignition temperature.

The curve for combustible of the 22-in. thick bed becomes horizontal at 18 in., showing that reactions have stopped and the gases are unable to take up more combustible; cessation of reaction is not caused by the incoming fuel, because this enters far enough above the 18-in. plane not to decrease the temperature at that plane. With the 22-in. bed the combustible per pound of air contained in the gases leaving the bed is 0.155 pounds.

With the 18-in. thick bed, abstraction of heat by the incoming fuel prevents the reactions from proceeding as far as with the 22-in. bed. The curve would be like that shown, and the gases might carry 0.152 lb. of combustible per pound of air.

Curves for other thicknesses are shown. The curves are not from experimental determinations but show the trend; the exact shapes will depend on the imposed conditions. The dotted curve through the extreme points gives the relationship between combustible per pound of air and thickness of bed that must obtain with equilibrium burning.

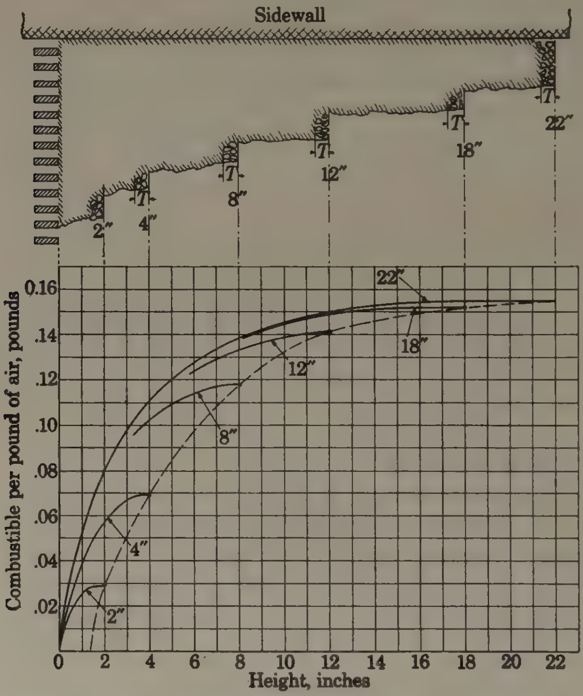


FIG. 6.—OVERFEED BURNING WITH RESTRICTED IGNITION. EFFECT OF THICKNESS OF BED ON COMBUSTION.

With constant primary air the rate of feed of the fuel decreases with decrease in thickness, but the weight of fuel to be ignited per pound of combustible burned is the same for all thicknesses. The heat radiated from the surface of the bed will depend on the temperature of the surface, and increases with decrease in thickness because there is less endothermic reaction of CO_2 to CO. As the bed becomes very thin there will be excess O_2 and no CO, with a resulting lower average temperature; the heat loss from the surface of the bed will reduce the temperature still further, so that the rates of reactions will be lower than at the same height of a thick bed; thus the rate of burning will be less than the rate of burning at the same height of a thick bed. The result of these actions will be that the rate of burning for various thicknesses will be less than those which would be predicted from the original curve for the 22-in. bed.

The foregoing has not considered the effect of the air rate. The rate of burning for a constant thickness would tend to be proportional to the air rate, except for the imposed conditions; because the heat lost to the side walls and by radiation from the surface of the bed do not increase as fast as does the heat developed, the temperature of the bed, and thus the rate of the reactions, increases with air rate, so that the rate of burning increases somewhat faster than the air rate, especially at the lower rates.

Considering the effect of air rate on ignition, the higher velocity of the hot gases over the surfaces of the pieces of incoming fuel will result in a somewhat higher rate of heat transfer to them. On the other hand, the rate of travel of the fresh fuel downward will be proportional to the air rate when the fuel fed is also proportional to it. Thus distance *T* will be greater with increase of air rate, and consequently the height over which reactions can occur will be less. This reduction will be of little importance with thick beds, but will affect the rate of burning with very thin beds.

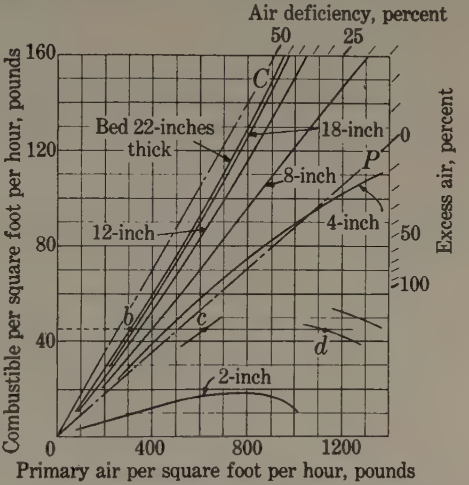


FIG. 7.—OVERFEED BURNING WITH RESTRICTED IGNITION. EFFECT OF RATE OF AIR SUPPLY.

A more important factor in thin beds is the effect of the relative sizes of the pieces of fuel to the depth. The burning will have to reduce the original volume of the pieces of fuel to that occupied by the ash in the short distance of the thickness of the bed, and the time in which this must be done will be inversely proportional to the air rate. As a result, pieces of fuel at the bottom of the bed will not be held down as well by the large pieces as they are in thick beds, and they will be more likely to be carried out of the beds with high air rates. Another practical factor is that holes are more liable to develop with thin beds with increase of air rate, with a consequent reduction in the average rate of burning.

As a result of these actions, the rate of burning will increase and reach a maximum with some air rate and thereafter fall off with further increase of air rate. Fig. 7 applies the same type of diagram as was used for underfeed beds and shows the effect of air rate for beds of various thicknesses; the shape of the curves is not based on direct experimentation but indicates the expected trends.

Fig. 7 can be used to show what will happen when the point of operation is moved. Suppose, for example, that it is desired to predict the effect of rate of primary air when operating at 45 lb. of combustible per hour; that is, to move horizontally on the line of 45 lb. of combustible. The minimum air rate that would give the 45 lb. would be point *b*, 310 lb. with a bed 22 in. thick; there will be about 30 per cent air deficiency. Moving farther along the line *bd*, small increases in air rate rapidly decrease the thickness of bed necessary to give the required rate of burning; also, it will be recognized that there is only one thickness for each air rate that will give the 45-lb. rate of burning. Operating at point *c* would give 20 per cent excess air, and the thickness might be 3½ in. At some higher air rate the line would cross a falling thickness curve at *d*, and further increase in air rate would necessitate an increase in thickness of bed.

There would be some limiting curve cutting through all the thickness curves that would give the air rate for each thickness that would carry some definite percentage of fuel out of the bed. Before this rate is reached, the loss of fuel pieces in the gases would increase gradually, and the form of plot would have to be changed. For each thickness curve as shown on Fig. 7, there would be a curve for the rate of feed associated with it; the vertical height between these two curves would increase with increase of air rate because of the increase in solid fuel in the gases.

The curves of Fig. 7 would apply to only one size of fuel; each size would have its own set of curves. Discussion of the plots for overfeed burning is not as satisfactory as it was for underfeed because the experimental data are lacking for plotting assured values.

High-temperature coke has been used for illustrating the principles for both underfeed and overfeed burning. The same principles hold for fuels with volatile matter as long as they do not cake. For caking fuels, the effect of the caking on the distribution of the air over the bed and on the size of the pieces of coked fuel must be included, and prediction of results is not nearly as certain at low air rates. The effect of caking at high air rates is less in underfeed burning, or with thin beds in overfeed burning. The scope permissible for this paper precludes treating these phases in detail.

Nor can crossfeed burning be discussed. It is of interest because it includes chain-grate stokers; ignition and burning on the crossfeed principle also occur in underfeed stokers.

STRUCTURE OF FUEL BEDS

If a fuel bed is burning in equilibrium with constant thickness it is evident that the structure throughout it will also remain the same. Each piece of fuel will undergo a definite series of changes in the order of heating, including evolution of gases, oils and tars for fuels containing volatile matter; ignition of the carbon; release of ash in proportion to the carbon burned; and, finally, the last trace of carbon disappearing at the end of the travel through the bed.

As stated previously, the ash released from a piece of fuel will travel with it and will not be affected by gravity unless it fuses and becomes fluid enough to flow downwards, no matter what the direction of the fuel. There are other factors that may affect the distribution of the ash in the bed; some of the particles of ash released may be carried along with the gases or may stick to other pieces of fuel; or some droplets of slag may be carried along (the latter is more likely to occur when gases are evolved inside the droplet, thus decreasing its apparent density). The uniform flow of the fuel can be disarranged by lumps of clinker sticking to the side walls or grates. The ash from different pieces of fuel may stick together to form one piece; this will not directly affect the uniform flow through the bed, but it may do so indirectly, because of the disturbance of the uniform flow of air and gases and the consequent nonuniformity of the burning.

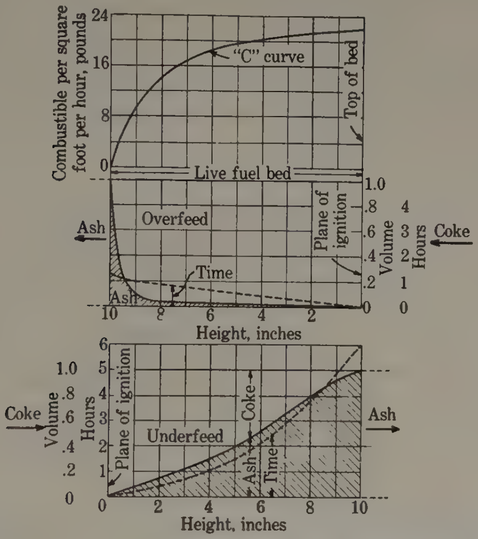
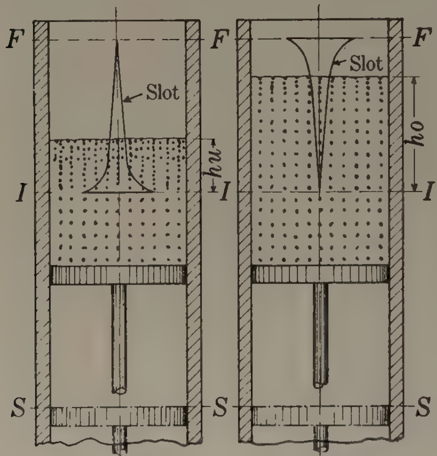


FIG. 8.—DISTRIBUTION OF ASH ON OVERFEED AND UNDERFEED FUEL BEDS BURNING HIGH-TEMPERATURE COKE.

In the U.S. Bureau of Mines *Bulletin* 378, calculations were developed to show the difference between the distribution of ash in underfeed and overfeed beds. It is simpler for illustrating the principles to assume that the fuel is high-temperature coke.

The top plot of Fig. 8 shows the rate of combustion, or what the author calls the "*C* curve." The left vertical axes are the bottoms of the live fuel of underfeed and overfeed beds of the types shown respectively by *A* and *B* of Fig. 1. The horizontal axes are the heights, and the tops of the beds are at 10 in.; that is, the live beds are 10 in. thick. The ordinates of the *C* curve represent the pounds of combustible carried in the gases at each height per square foot of grate area per hour. The burning for both beds is assumed to be in equilibrium, and no combustible passes to the ashpit; therefore, the value of *C* at the top of the bed equals the weight of combustible in the coke fed per square foot per hour.

The other values assumed—but which have no particular interest in this discussion—are: Coke fed per hour = 25 lb. per sq. ft. per hour; apparent density of coke pieces = 30 lb. per cubic foot; percentage of combustible in coke = 88; percentage of ash = 10; percentage of ash leaving the bed = 20. The apparent density of the ash and clinker in the bed is assumed to be 30 lb. per cubic foot; this value is taken low to include the hollow spaces in the beds instead of assigning them to the coke.



A Underfeed *B* Overfeed (upside down)

Fig. 9.—Illustration of differences between underfeed and overfeed fuel beds.

The middle plot of Fig. 8 is for an overfeed bed. The dotted one is the time needed for a piece of fuel to travel the given distance down the bed. The full curve shows the proportion of the volume of the bed occupied by the ash; thus if the ash occupies 0.10 of the volume the remaining 0.90 of the bed is occupied by coke. The bottom plot shows the same values for the underfeed bed.

It must be remembered that these plots assume similar beds with the same rate of burning. However, this does not mean that the size of the coke pieces is the same; in fact, they would have to be smaller for the underfeed bed if the *C* curves are to be the same.

The plots show the great differences between the times of travel and the ash distribution in underfeed and overfeed beds. The average time required for a piece of coke to travel through the live fuel bed is 6.0 hr. for the underfeed and 1.35 hr. for the overfeed, or 4.5 times as long. The percentage of the whole volume of the live fuel bed occupied by the ash is 44 for the underfeed and 3.1 for the overfeed, or 14 times as great for the underfeed.

The differences between the two types of bed are illustrated in Fig. 9. Each view depicts a cylinder fitted with a piston. The wall of each cylinder has a slot cut through it; these slots are the same shape, except that *A* has the broad end at the bottom and *B* at the top; the slots are covered by fine screens. Above the pistons is water containing pieces of sponge, uniformly distributed and having the same specific gravity as the water, which are assumed not to move horizontally except as they are crowded aside by other pieces of sponge. Originally the pistons were at level *SS* and the top of the water at *II*. The pistons were moved upwards at the same constant rate to the positions shown. It is evident that more water will have run out of the larger area of the slot of *A* than out of *B*, and that consequently the top surface of the water of *B* will be higher than that of *A*. The sponge—representing the ash—will therefore be packed more densely in *A* than in *B*, and its distribution will be as depicted. Ultimately the top of the water in *B* will reach the top of its slot *FF*, and it will not rise any higher if the rate of movement of the piston is regulated so that the rate of supply of water is equal to that which runs out of the slot. After another interval of time, the surface of the water of *A* will reach *FF*, but it will take considerably longer because for all heights before reaching *FF* the rate of losing water through the slot of *A* will be faster than through that of *B*. If the effect of the head of the water on the rate of flow is eliminated, the rates of flow through the slots will be the same when the surface of the water of both cylinders is at *FF*, and a rate of movement of the piston could be chosen so that the water would not rise farther and there would be an equilibrium condition in each cylinder, the pieces of sponge piling up on the top of the water above the level *FF*. After that, every piece of sponge would take the same time to travel from *II* to *FF* as did initially the surface of the water as it rose the same distance. These movements correspond to the actions in underfeed and overfeed beds as depicted by Fig. 8; the widths of the slots are the ordinates of the *C* curve.

The calculation used to determine the distribution of the ash can be extended to show what happens to the coke pieces as they progress through the bed. There are three things of interest, the velocity of the pieces, their size, and the number of pieces in each plane. Fig. 10 shows three curves both for an overfeed and an underfeed bed, one for the velocity of the pieces as a proportion of the velocity of the coke at the plane of ignition, one for the linear size of the pieces as a proportion of the linear size of the coke as fed, and one for the number of pieces of coke as a proportion of the number at the plane of ignition. Comparison of the velocity curves shows that the average velocity over the length of the bed is much less for the underfeed than the overfeed. The speed of the pieces of the overfeed does not change much until the greater portion of the length has been traveled, whereas in the underfeed it falls off rapidly at the start. The velocity at the end of the travel in both beds is not zero, but is that of the ash as it leaves the bed; also, this velocity is the same for both beds—as it must be with the same rate of feed.

Similarly, for the size of the pieces; in the overfeed the rate of decrease is very slow until the piece approaches the grates, when the change is rapid; in the underfeed the change in linear size is fairly uniform over the height of the bed. It should be noted that the curves are for linear size; the curves for the volume occupied by the pieces would show a still greater difference between the two types of bed. By "linear size" is meant the linear size of the space occupied by each piece of coke, which can be considered as a cube. The number of pieces in the various planes is derived from the size of the pieces and the volume occupied by the ash. The two curves for the number of pieces have not been extended to the axes; the original *C* curve chosen for illustrating the ash distribution does not truly represent equilibrium; theoretically it should be tangent to the horizontal and small changes materially affect the size and number when the pieces are near the vanishing point. For the *C* curve as shown in Fig. 8 the pieces of the underfeed bed would not be completely burned out, or there would have to be a sudden change in the laws for the rate of reaction.

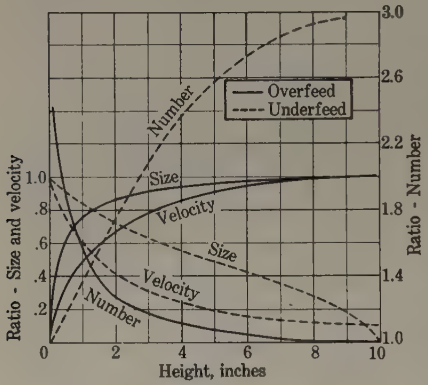


FIG. 10.—VELOCITY, LINEAR SIZE AND NUMBER OF PIECES IN A PLANE AS RATIOS OF VELOCITY, SIZE AND NUMBER OF PIECES RESPECTIVELY AT PLANE OF IGNITION. HIGH-TEMPERATURE COKE.

It has been assumed that each piece of fuel was originally the same size and that the rates of burning are also the same. These assumptions are not necessary, and as long as the ratio of carbon to released ash is maintained at that called for by the computations the sizes can be dissimilar.

The curves of Figs. 8 and 10 give all the data required for drawing a section through the beds and for showing the condition and distribution of the coke pieces and released ash. The difficulty at once arises as to how to fit in the pieces and at the same time satisfy the mathematics, which assume that the spaces occupied by the coke and the ash are plastic, so that they can satisfy the assumed conditions that the apparent density of each remains constant through the bed. The difficulty can be avoided in a number of artificial ways, such as juggling the voids between pieces while maintaining the total called for or assuming that some pieces are burnt more than others. The more correct solution is the certainty that, although we can speak of a constant rate of burning and imagine a *C* curve that does not change with time, in reality such assumptions can be correct only as the average for a period; although it might be true for successive 15-minute periods, yet during each period there would be, even theoretically, cycles of change because of the discrete size of the pieces. Nevertheless, it is possible, with a little trouble, to draw a section through a fuel bed that will closely comply with the mathematical requirements; moreover, such a representation would not be merely a theoretical fancy but could show actual conditions fairly closely.

Although the time required for a piece of coke to pass through an underfeed bed will always be much longer than in an overfeed, it is improbable that one could duplicate exactly the distribution shown by the curves of Fig. 8, because normal ashes would fuse and bury the small pieces of coke, especially in the underfeed, and they would not have an opportunity to burn; the result would be that combustible would be carried out with the clinker. However, one can represent such a condition because the curves can be stopped at a height that would leave any percentage of combustible desired as unburned; this stoppage would not affect the values for the bed below that plane. In practice one would usually clean the bed before it became clogged with such a large proportion of clinker.

The average values given by the calculation are a necessity if one would proceed further and show that the results satisfy certain laws of rates of reaction or, on the other hand, if one would deduce the laws from the results. However, although the average volume or linear dimension of the coke pieces in any one plane is given, there might be some question on the average shape of the pieces and thus on the surface area, although it is probable that any assumption made for the shape will introduce less error than would be associated with application of the laws of reaction, the temperature and the conditions imposed on the bed by the side walls and other causes of heat loss, and uneven burning over the area of the bed.

The Future of Coal for Stationary Power

By E. H. Tenney*

(St. Louis Meeting, October, 1935)

A discussion of the probable future use of coal for power development involves the study of several basic factors, such as future demand for power, the quantity and availability of fuels in direct competition with coal, the price of coal, and a closely related factor—the efficiency and cost of fuel utilization.

Predictions based on purely statistical analyses of past coal consumption have definite limitations. For example, statistics showing the relation between coal utilization and total energy consumption from all sources between the years 1890 to 1930 indicate that during that period the energy consumption per person in the United States increased five times, while the coal consumption per person increased only twice[1], and there has been an actual decrease in per capita coal consumption since 1920.

Figures such as these, compiled by authentic agencies, emphasize the need for caution in attempting estimates as to the future use of coal, and, incidentally, would seem to offer little basis for optimism in the coal industry. They also suggest the value of analyzing broadly the part played in the past by the related factors of power consumption, competitive fuels, coal prices and the efficiency of utilization. Proper weight must also be given in such a study to the present status and limitations of power-plant developments, and to the question of what the coal industry itself can do to further its own interests and influence the consumption of its product.

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[1] W. S. Hutchinson and A. J. Breitenstein: *Trans.* A.I.M.E. (1934) **108**, 461–466. *Per capita* energy and coal consumption in the United States:

	Population, Millions	Energy Consumed, Million B.t.u.	Coal, Tons
1890	62.9	16	2.49
1910	92.0	47	5.31
1930	122.8	73	4.21

In the consideration of such things we may hope to arrive at some sort of a reasonable basis upon which to predict the future use of coal for stationary power.

Demand for Power

While it may not be reasonable to assume that the same rate of increase in power consumption in the United States will continue in the immediate future as has prevailed in the past[2], still the indications for a continuation of this increase are certainly encouraging. Probably the best indicator, at least the one most available, is the per capita growth in electric power consumption during the past two years. A growth curve throughout the entire United States is shown on Fig. 1[3]. An interesting feature is the continued parallel growth of industrial, commercial and domestic power demand up until the beginning of the 1930 depression. Then in spite of the falling off in both industrial and commercial power sales, the domestic load has continued to increase throughout the past five years until now the total electric power consumption has reached a new maximum. The reasons for this unprecedented growth in domestic load are many and range all the way from decreased rates to concerted sales efforts, and to the development of home appliances. It seems reasonable to expect similar increases in the industrial and commercial loads along with the next upswing in business, and when such increases

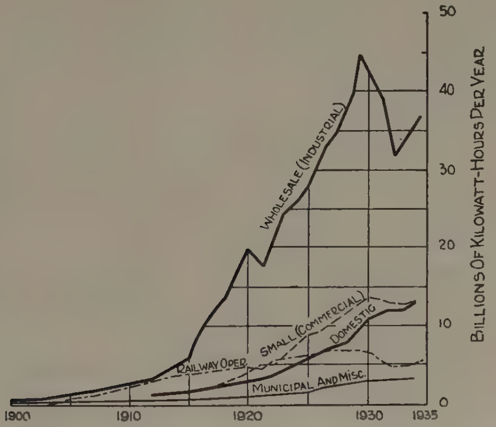

Fig. 1.—Growth in use of electricity in United States, by class of consumer.

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[2] Reference of footnote 1, 464, which shows that energy supply in 1850 was equivalent to 47 trillion B.t.u.; in 1870, equivalent of 208 trillion; 1890, equivalent of 1049 trillion; 1910, equivalent of 4341 trillion; 1930, equivalent of 9031 trillion; and in 1950, by extrapolation, equivalent of 14,500 trillion B.t.u.

[3] *Electrical World* (Jan. 5, 1935) **105**, 36.

come the curve of per capita growth in power demand may well continue the sharp increases of the past two years.

Competitive Sources of Power

While the per capita consumption of power has been steadily advancing, the portion of that power generated by coal has been steadily decreasing, and the increased efficiency of utilization has further reduced the proportionate consumption of coal[4].

Whereas over 90 per cent of all the forms of mechanical power consumed in this country in 1900 was derived from coal, with water power and petroleum products dividing the remainder, in 1930 coal accounted for only 60 per cent, hydro power had advanced to 8 per cent and petroleum products to over 30 per cent.

The reasons for this situation are found only partly in the higher cost of coal, for equally as important are the quantities of oil and hydro power available, and the trend to highly organized technical development in the gas and Diesel engine, in oil firing, in the distribution of oil and gas through pipe lines, and in the transmission of electricity from distant sources of water power to points of consumption. These trends in technical development and these competitive sources of power furnish the clue to the future of coal for stationary power.

Petroleum products will doubtless be confined to the internal combustion engine and therefore need not be seriously considered for large steam-raising purposes except under exceptional circumstances. Neither oil nor gas can compete with coal on a "cents per million B.t.u." basis in the greater part of the country, and their use under boilers is economically unsound except for unusual conditions such as temporary arrangements where the steam-raising equipment is in too poor a condition to properly burn coal, or when excess gas must be dumped pending more logical markets. It is also significant that compared with modern coal-firing methods, under all but very small boilers, oil and gas are inherently at a disadvantage because of the hydrogen losses in their combustion.

Developments in the field of the stationary gas engine are practically at a standstill, except possibly in the oil fields, where there have been some recent gains in small gas-engine installations for isolated power purposes. Except for purely local conditions, however, and for all practical considerations the Diesel engine should be the only petroleum products competitor with coal for stationary power, but no appreciable improvement in their economy has been effected since Dr. Diesel built his first engines some 40 years ago, whereas there has been great progress in coal and steam economy improvements. The cost of Diesel engines is high. A Ford V-8 engine delivering about 60 hp. continuously can be considered as selling for $300; that is, $5 per horsepower[5]. Diesel engines cost more than eight times as much. Steam plants increase rapidly in efficiency as they increase in size, but the small Diesel is practically as economical as the large, and the cost per unit of capacity is almost the same. It seems unlikely that any but the smallest coal-fired steam plants should face any competition from the Diesel plant.

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[4] Reference of footnote 1, 462.

The specter of diminishing supply of oil and higher cost faces the oil industry. The latest semi-authoritative study[6] of this situation indicates known recoverable oil reserves in the United States to be 12 billion barrels, enough for only 13 years at the 1934 consumption rate. Particularly significant is the 1931–1934 slump in the annual rate of discovery of new oil. In that period new wells "brought in" produced an average of 580,000,000 bbl., against a discovery rate during the years 1926–30 of 1,990,000,000 bbl. Indeed, the situation may some day be reversed, with coal being used for supplying motor fuel. It has been estimated[7] that the available coal in this country within 3000 ft. of the surface would yield 92,000,000,000 bbl. of motor fuel, over seven times the available supply of petroleum oil. It might be mentioned also that another threatening source of competition to petroleum products in this field seems to be found in the recent development of the small automatic gas producer.

In the rapidly growing central-station (electric utility) industry the fuel consumption of the past 15 years among fuel oil, gas, coal and hydro power is distinctly interesting (Table 1)[8]. It is significant that whereas the efficiency of utilization increased from the point where 3.2 lb. of coal was required per kilowatt-hour in 1919, to 1.45 lb. in 1934, the total consumption of coal remained reasonably constant because of greater kilowatt-hour generation, and ranged between 30,000,000 tons in 1921 and 1933 to 44,000,000 in 1929. *At the same time consumption of oil* for stationary power has decreased sharply since 1924. Gas consumption for the same purpose has increased since 1929 to the point where its equivalent total heat content per year is almost twice that of oil although only one-seventh that of coal. As suggested before, such gas firing seems to be specifically a depression, distress-market situation.

Coming to hydroelectric power (Table 2 and Fig. 2) it is impressive to note that since 1919 there has been little change in the relative position between coal and hydro plants, each sharing about equally in the growth in electric generation. Fuel[9] in both 1919 and 1934 accounted for

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[5] E. J. Tangerman: *Power* (June, 1934) **78**, 367.

[6] Goodbody and Co., New York (June 17, 1935, monthly letter).

[7] Handbook of Petroleum, Asphalt and Natural Gas, Rev. Ed., 1931.

[8] U. S. Geological Survey. Monthly report (Apr. 20, 1935) on production of electricity for public use in the United States in 1934.

[9] G. A. Orrok: *Power* (June, 1934) **78**, 363.

62.5 per cent of the generation while hydro plants carried 37.5 per cent. As to the future of hydro power, however, there is the possibility of a limit in available capacity. Orrok has estimated that the potential water power in the United States, utilizing every source that could conceivably be developed for $500 per kilowatt, may represent a total of 32,000,000 kw. available 50 per cent of the time, or an output roughly 150 billion kw-hr. per year. This is the maximum output of hydro power that there is any possibility of developing, but at current coal prices development to this extent would by no means be justified. The electric-power industry alone has today an installed capacity of 36,000,000 kw.[10], of which 10,000,000 kw. is hydro. It is generally believed that the development of many water-power projects today, particularly under political subsidy, is not economically sound. In the future, especially if the relative position of price and technical methods remains the same, there will doubtless be fewer hydro plants constructed. As George A. Orrok has said, "It will probably be many years before our present installation of 10,000,000 kw. of hydraulic power will be notably increased."

TABLE 1.—*Annual Consumption of Fuel in the Production of Electricity for Public Use in the United States*, 1919–1934

Coal			Fuel Oil		Gas	
Year	1000 Short Tons	Change from Previous Year, Per Cent	1000 Bbl.	Change from Previous Year, Per Cent	Million Cu. Ft.	Change from Previous Year, Per Cent
1919	35,100		11,050		21,406	
1920	37,124	+ 5.8	13,123	+18.8	24,702	+15.4
1921	31,585	−14.9	12,045	− 8.2	23,722	− 4.0
1922	34,179	+ 8.2	13,197	+ 9.6	27,172	+14.5
1923	38,966	+14.0	14,684	+11.3	31,433	+15.7
1924	37,556	− 3.6	16,630	+13.3	48,443	+54.1
1925	40,222	+ 7.1	10,246	−38.4	46,521	− 4.0
1926	41,311	+ 2.7	9,399	− 8.3	53,207	+14.4
1927	41,888	+ 1.4	6,782	−27.8	62,919	+18.3
1928	41,350	− 1.3	7,158	+ 5.5	77,326	+22.9
1929	44,937	+ 8.7	10,124	+41.4	112,707	+45.8
1930	42,898	− 4.5	9,260	− 8.5	120,290	+ 6.7
1931	38,734	− 9.7	8,123	−12.3	139,328	+15.8
1932	30,290	−21.8	7,967	− 1.9	107,875	−22.6
1933	30,575	+ 0.9	9,940	+24.7	102,601	− 4.9
1934	33,555	+ 9.7	10,379	+ 4.4	127,896	+24.6

In this connection, the capacity of hydro projects can bear close scrutiny. The vagaries of rainfall are well known. Power that is available 50 per cent of the time will not suffice for most industry, and hydro plants are confronted with the necessity for steam-plant standby. There seems to be little doubt that in the future the combined effect of high investment for water power and an additional investment for partial steam standby will receive careful consideration before hydro developments are projected.

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[10] Reference of footnote 9.

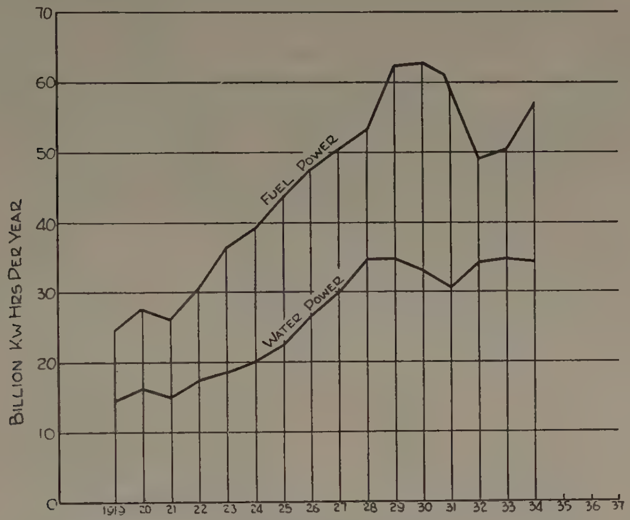

Fig. 2.—Annual electric energy production for public use by power plants in United States.

Fuel Price and Efficiency of Utilization

To a perhaps unappreciated extent, the efficiency at which coal is utilized depends upon its price. We have seen the effect of a sharp increase in coal price quickly followed by a marked increase in fuel economy in the recent war period. The rise in weighted average coal price between 1916 and 1917 from $1.25 to $2.60 per ton increased the fuel bill of consumers[11] from $672,000,000 to $1,388,000,000. This acted as a tremendous stimulus to power users to modernize and improve the economy of their plants, and whereas the fuel rate per kilowatt-hour average throughout the country in central stations had in the previous decade decreased from perhaps 4 lb. of coal to 3.2 in 1919, by 1927 this had become 1.69 lb. From that point improvement has been slight, although it moved to 1.45 in 1934. (See Fig. 3.)

The best fuel consumption for stationary plants was achieved about five years ago—about 1.2 lb. coal per kilowatt-hour. Most power men feel that this level is all that is economically justified, that the stress should no longer be placed on extremely high economies but on developing simplicity of operation, low investment and low maintenance costs.

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[11] A. T. Shurick: *Trans.* A.I.M.E. (1934) **108**, 469.

The chief means for improving economy in electric generation during the '20s was the introduction of the regenerative cycle for boiler feed-water heating, which enhanced the value of higher steam pressures. With 1400 lb. now the generally accepted top, no developments of the straight steam cycle that promise any great change are on the horizon, and it is just possible that we may now be seeing the end of sharp improvements in fuel economy; however, competition from other fuels and the use of binary vapor cycles definitely relieve the threat of reduced coal consumption with rising prices.

TABLE 2.—*Annual Production of Electricity for Public Use in the United States,* 1919–1934

Total		Water Power		Fuel Power	
Year	Million Kw-hr.	Million Kw-hr.	Per Cent of Total	Million Kw-hr.	Per Cent of Total
1919	38,921	14,606	37.5	24,315	62.5
1920	43,555	16,150	37.1	27,405	62.9
1921	40,975	14,970	36.5	26,005	63.5
1922	47,654	17,207	36.1	30,447	63.9
1923	55,665	19,343	34.8	36,322	65.2
1924	59,014	19,969	33.8	39,044	66.2
1925	65,870	22,356	33.9	43,514	66.1
1926	73,791	26,189	35.5	47,602	64.5
1927	80,205	29,875	37.2	50,330	62.8
1928	87,850	34,696	39.5	53,154	60.5
1929	97,352	34,629	35.6	62,723	64.4
1930	95,936	33,021	34.4	62,915	65.6
1931	91,729	30,603	33.4	61,126	66.6
1932	83,153	34,098	41.0	49,055	59.0
1933	85,402	34,727	40.7	50,674	59.3
1934	91,150	34,058	37.4	57,092	62.6

In what direction then should the coal industry expend its efforts to assist in maintaining coal as a primary source of power in the future? I think it should look to a better technical understanding of the power plant's problems. For smaller plants, in particular, the coal industry can actually take the initiative in showing operators how to operate more cheaply and to better advantage with coal, whether it be along the lines of the continued use of old equipment, or an increased rate of output through more effective methods of operation, or any other way that will tend to preclude the encroachment of competitive fuels. This will require a technical knowledge of the factors that determine not only fuel economy in boiler rooms, but the operating expense, the maintenance, and the cost of the coal-burning equipment, all of which will determine the competitive position of coal.

It is fitting to mention here a few of the items, other than delivery of tonnage, concerning which the mine operator should cooperate with the consumer and about which the coal salesman should warn his customer, and thereby promote the consumption of his product by helping to reduce the *cost* of utilization.

1. A factor that might well engage the attention of the coal men interested in modern power-plant operation is the amount and the composition of the ash content of bituminous coals. Furnace heat liberations will vary from 12,000 B.t.u. per cu. ft. per hour to 60,000, depending on the tendency of ash to clinker on stokers or to plug up the gas passages between the boiler tubes in pulverized-coal firing.

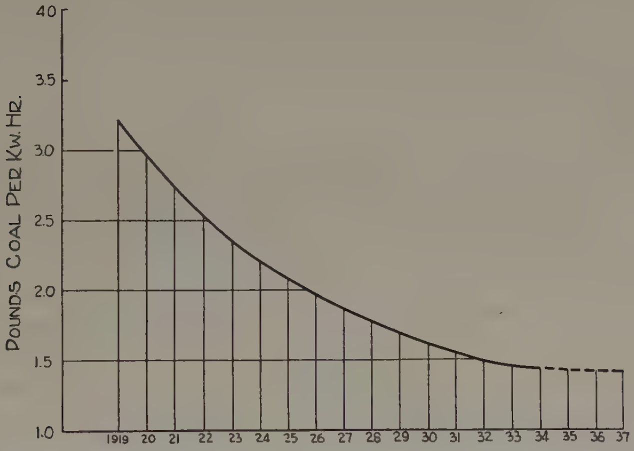

FIG. 3.—AVERAGE POUNDS COAL PER KILOWATT-HOUR FOR ALL CENTRAL STATIONS IN UNITED STATES.

The importance of the fusing tendency of ash in coal firing assumes greater importance when we realize that furnace volume and furnace investment costs may be greatly reduced if the quality of the fuel to be used is definitely known by the designer, and if the men responsible for coal supply will conscientiously see to it that inferior grades of coal are not supplied. Here is a point on which the coal-mine operator, and indeed the miners, can, by some control over their product, assist the consumer in getting a grade of coal that will give a maximum heat release in his furnace and result in less initial cost, less maintenance cost, and a maximum furnace capacity.

2. In plants using pulverized fuel another coal factor is that of the expense of grinding and the fineness of grind possible with a given fuel. This factor follows from coal hardness, the tonnage to be prepared for

a given heat content (ash having no value) and the tendency of moisture combination with such material as clay to become sticky and plug up pulverizing mills.

3. Water treatment both before and in the boilers can effect considerable savings. Scale causes a loss in boiler capacity and is expensive to remove. This expense as well as the loss in efficiency, the damage due to corrosion and priming, the loss in running capacity, and the periodic outage of equipment, could all be avoided by properly treated and controlled boiler water.

Finally, coal operators in meeting the pressure of other fuels and the burden of a higher price on their own can advisably watch developments in the art of firing and keep in mind the status of their own particular coals as it is affected by such developments.

Summary

In this discussion it has been shown that the outstanding trends and forces that shape the future of coal in the field of stationary power are:

1. The growth of per capita power consumption in this country. The encouraging growth in the electric utility field of the domestic power and light load all through the depression should be shortly accompanied by a recontinuance of the growth in power consumption by industry and commerce. While it is difficult to say what this growth will be quantitatively, the growth of power consumption in the past indicates it should be substantial.

2. The position of competitive power sources, especially hydro power and petroleum products. Whereas five times as much power per capita is now being consumed in this country as was being used 40 years ago, the coal consumption per capita has less than doubled in that period, and has actually shown a per capita decrease since 1920. This has been partly due to the development of hydro power but much more to the development of the internal combustion engine and petroleum industry.

3. The amount of future hydro-plant development is absolutely limited to much less than present steam-plant capacity. In fact, other than under political subsidy, the indications are that hydro developments are close to true economic saturation today.

4. The visible supply of petroleum oil in this country, according to the most recent estimate, is limited to 13 years at current consumption. Particularly significant is the 1931–1934 slump in the annual rate of discovery of new oil sources.

5. The price of coal and the efficiency of utilization seem closely related. An increase from $1.25 to $2.60 late in the World War was subsequently followed by an unprecedented increase in the efficiency of burning coal; in the electric generating plant the coal rate per kilowatt-hour decreased from 3.2 lb. in 1919 to 1.69 in 1929.

6. The increasing efficiency of coal burning seems to be flattened out at present, with boiler efficiencies not likely to exceed the maximum of 90 per cent, and the fuel rate in electric generating plants not falling below 1.25 lb. per kilowatt-hour. There is no assurance, however, that an increased fuel price would not develop means for further improvement.

7. There are several known ways in which the coal-mine operator can assist the power industry to use more coal, by reducing the *cost* of utilization. Examples given are: (*a*) reduction of ash and refuse content, and particularly control of its fusing temperature so as to permit higher furnace-heat release and consequently lower investment and operating costs; (*b*) control of refuse and moisture content to facilitate coal preparation in plants using pulverized fuel; (*c*) elimination of boiler scale.

Conclusion

It seems probable that there should be a marked growth in coal consumption for stationary power purposes, in view of the expected growth of per capita power consumption in this country, which should tend to increase fuel consumption; of the limitations to the supply of petroleum products such as oil and natural gas, as competitive with coal; of the apparent economic saturation, barring political subsidy, in hydro plant construction; of the flattening out in the rate of increased efficiency of coal utilization as seen in electric utility central stations; and in view of the possibilities of reducing the cost of coal burning.

DISCUSSION

(*M. M. Leighton presiding*)

J. E. Tobey,* Cincinnati, Ohio (written discussion).—Mr. Tenney's paper truly reflects the good judgment and clear vision of the author, and is a genuine contribution to coal and power interests. The following is a discussion of the three natural divisions of the paper:

Demand for Power

The increasing mechanization of factory, farm and home, due to availability, lower cost and broadening adaptation of electric energy will insure an increasing demand for power. The fact, as pointed out in the paper, that domestic load increased consistently through the depression years, forecasts post-depression acceleration of sales of domestic electrical appliances and corresponding increase in the use of electricity. The Government report on electrical appliance sales shows that the dollar value of the appliance sales increased 22 per cent during the first six months of the year 1935 over the same period in 1934. Gross sales increased from $24,764,193 to $30,349,676. Favorable to this situation is the fact that there has been little, if any, increase in electric rates to domestic consumers. According to Roger Babson, fuel and light costs have been increased only 2 per cent since the depression low (June, 1933) whereas there have been large increases of other items, for example, food 35 per cent, clothing 22 per cent, housing 14 per cent, etc. Sales resistance to electrical appliances is very low because of the appeal of their labor saving, convenience, and comfort factors. The public is intrigued by the miracle-like accomplishments of these appliances, and will adopt them as fast as finances will permit.

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* Manager, Fuel Engineering Division, Appalachian Coals, Inc.

The industrial electric load, which fell off heavily during the depression, is showing a rapid increase. This is in part contingent upon the demand for and manufacture of domestic appliances, but more so to the pick-up in the basic industries. We already see the beginning of a revolutionary demand for new and greater electrically operated machines and related equipment to replace and augment those that are worn out and obsolete. This, together with the continual development of new processes requiring electrical power, seems to definitely insure growth of industrial power. All other types of electrical load are following these same trends at the present time. The United States Geological Survey reports that "total production of electricity for public use in the United States in the month of August (1935) was the largest ever produced in any previous August." The average for the month was 277,000,000 kw. per day.

Competitive Sources of Power

The writer concurs in Mr. Tenney's belief "that petroleum products doubtless will be confined to the internal-combustion engine, therefore need not be seriously considered for large steam-raising purposes except under exceptional circumstances." These fuels will react to the law of diminishing supply and gradually will withdraw from the stationary power field. The use of fuel oil is now on the wane in stationary plants. At the present time there is a flair for natural gas for stationary power use, which undoubtedly will subside with the increase in demand for the more profitable uses of this gas, and when the effects of a diminishing supply are felt.

The writer feels that the Diesel engine is destined to be short-lived as a source of stationary power. It appears now that the Diesel engine has a place in automotive transportation, and that it will replace a large portion of the present equipment driven by gasoline engines, especially in trucks, tractors and buses. Therefore, if the Diesel becomes popular in that field, the economic factors, causing a marked increase in the price of fuel, will automatically remove it as a competitor of coal in stationary power practice.

The large hydroelectric developments and the proposed similar developments, all under Government subsidy, appear on the surface to offer a serious threat to coal-burning plants. A study of the map of the United States showing the great industrial areas and also the locations of these hydro sources of power reveals that they are not properly oriented so as to furnish economically hydroelectric power to these industrial areas. This is due to the fact that transmission of electrical energy over long distances is unprofitable. Had these projects been economically sound and profitable, they would have been developed by private interests in the past. Accordingly, unless the Government actually subsidizes transmission of power, or unless there takes place a shift in industrial centers, the latter being improbable, the great bulk of industrial power will continue to be coal-steam generated.

Fuel Price and Efficiency of Utilization

It is very difficult to predict future prices of coal because of the uncertainty of the effect of legislation and other unforeseen forces. Temporarily, all factors point to a slight increase in coal prices, but present trends indicate that coincidently there will be an increase in coal quality, as evidenced by the increasing number of cleaning plants being installed at the mines. Such increases will be limited, fortunately, for the coal consumed, by the surplus availability of competitive fuels in which labor costs are not a vital component of their total costs.

As mentioned by Mr. Tenney, efficiency of coal utilization in large power plants is probably near the top, and will be considered as such for some time to come, and that emphasis will be placed on developing simplicity of operation, low investment, and low maintenance costs. Further, the suggestion that much can be done to improve the efficiency, and general satisfaction in the use of coal in the smaller plants is very true and timely.

Inasmuch as fuel is by a wide margin the greatest item of expense in these plants, and because the smaller plants are not supervised by well trained power engineers, the writer feels that the coal industry, for its own protection, must fill the breach by giving such plants adequate engineering service. If the coal industry does not render this service it will lose considerable tonnage in the immediate future to competitive fuels whose forces are well organized and include in their program complete engineering service and counsel to their customers, to enable the customer to obtain the maximum benefits from the use of their product. Its representatives, properly qualified for this work, should see that the proper application of coal is made to these plants, and that the operators are properly instructed in its efficient use. They should be prepared to offer advice and counsel to the plant management in regard to the plant design and selection of equipment, and to assist plant operators in ironing out other operating difficulties that may exist in the plant, many of which are directly or indirectly associated in some way with fuel. The supplier of coal should render this service as a necessary obligation to a mutually satisfactory contract for his product.

Another matter worthy of consideration is the efficiency of power utilization itself. While large and expensive hydroelectric projects are being launched, which will provide more electric energy, we are confronted with the paradoxical situation wherein only about one-third of the present capacity of electrical generators is being utilized. It is true that generators are fairly well loaded for 8 or 9 hr. during the day, but night loads are very light and maintain for 15 to 16 hr. of the 24-hr. day. These generators are like the proverbial horse; they are standing idle a greater portion of the time eating their heads off—devouring fixed charges. Interest, depreciation, taxes, insurance and obsolescence work overtime or straight through 24 hr. a day. The economics of the situation dictate that the power equipment to which these factors apply should operate as nearly full time as is practicable. Man's aversion to night work should no longer stymie the development of night load or off-peak power. Processes that require large amounts of electrical energy but very little labor are forging ahead, such as electrochemical, electrometallurgical, and others, for the manufacture of synthetic products.

Inasmuch as the fixed charges for stationary power plants are borne by the original load and peak load for which they were designed, any new off-peak load that may be developed can be considered to be fixed-chargeless and can be generated for practically the fuel cost alone. The latter is in the neighborhood of from 2 to 3 mills per kilowatt-hour.

The following information and data will serve to prove that there is available today a vast pool of cheap electric power now unused, which may be had for the mere throwing of a switch.

The total capacity of electric generators installed in utility and industrial plants in the United States as reported to the Federal Power Commission[12] and the U. S. Census of Manufacturers (1929, latest census), totals approximately 40,000,000 kw. Of this amount approximately 32,000,000 kw., or 80 per cent, represents the installed capacity in the utility plants. Approximately 8,000,000 kw., or 20 per cent, represents the installed capacity in industrial plants.

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[12] Federal Power Commission National Power Survey—Interim report—1935 Power series No. 1.

Referring to the utility plants alone, the installed capacity is divided as follows: Approximately 23,000,000 kw. in fuel electric plants; approximately 9,000,000 kw. in hydroelectric plants. Calculations based on figures included in the Federal Power Commission Report, show that in 1933 the utility plants had a ratio of annual production to total output capacity of (only) 28 per cent (this included both hydro and fuel electric plants). Segregated, the output factor for the hydro plants was 40 per cent and that of the fuel electric plants was approximately 23 per cent. The figures on Diesel electric plants that were included in fuel electric plants when separated showed that Diesel engines generated less than 0.1 per cent of the electricity and had an exceptionally poor capacity factor of less than 2 per cent, which indicates that the

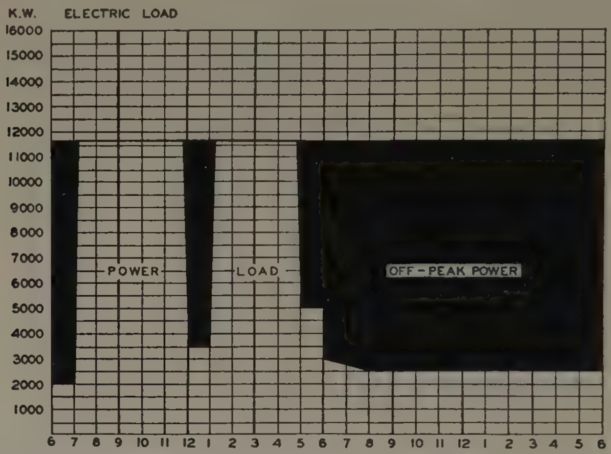

FIG. 4.—POWER LOAD OF TYPICAL LARGE INDUSTRIAL PLANT.
Dark areas represent off-peak power capacity that is not being used.

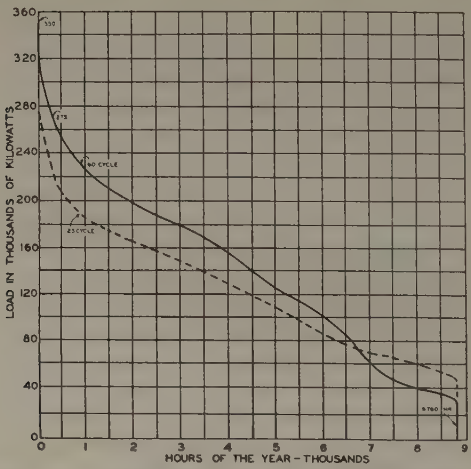

FIG. 5.—COMPOSITE LOAD CURVE FOR YEAR 1922, COMMONWEALTH EDISON CO. This is one of the most heavily loaded utility systems.

Area above curve, which apparently is at least 60 per cent of total capacity represents idle generator capacity in that system.

utilities must have operated but few of their Diesel plants. The production figures for the generators installed in industrial plants are not available, but the writer believes that were they available and the output factor developed, it would reveal a still lower figure than that for the utilities (which was 28 per cent).

The charts accompanying this discussion (Figs. 4, 5 and 6) typically illustrate idle generator and off-peak power capacity and the economies involved.

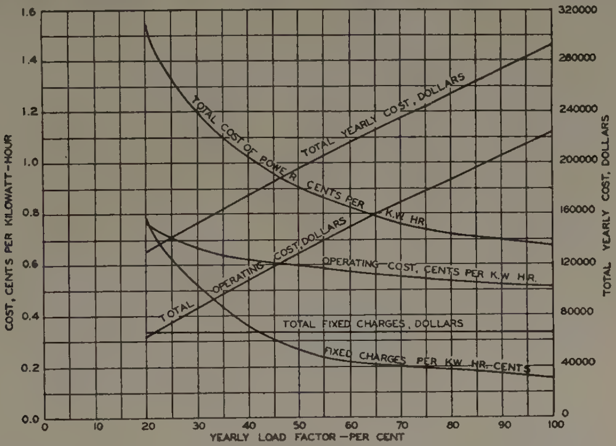

FIG. 6.—INFLUENCE OF LOAD FACTOR ON COST OF POWER AT SWITCHBOARD (MAXIMUM LOAD 5000 KW.).

Curve for fixed charges is flat and for operating costs nearly so. Curve representing total cost of power in cents per kw-hr. shows that in this particular plant an increase in load factor from approximately 25 to 90 per cent cuts the cost of power in half (from 1.4 to 0.7 ¢).

CONCLUSION

Trends indicate a continued increase in the demand for power; that coal will gradually displace the other fuels for stationary power; this gain will be partly offset by improved efficiency in industrial power plants, but this loss may in turn be overcome by the production of more off-peak power for new purposes. Government-sponsored hydroelectric power, because of its isolation with respect to markets for power, unless uneconomically exploited, must develop new uses for its output.

Coal tonnage for stationary power should increase, perhaps slowly in the immediate future, but accelerate more rapidly as the present disturbing factors are eliminated, after which it should increase more or less unhampered for a considerable period until the present methods of steam and electric generation are revolutionized and superseded by equipment that will operate on a much higher plane of over-all efficiency.

A. J. JOHNSON,* Primos, Pa. (written discussion†).—Two factors that Mr. Tenney mentions briefly would seem to deserve some amplification. First, even with an almost inconceivable complete political subsidy of our water-power developments, a serious and almost insurmountable problem of the location of the hydropower projects remains. It has been estimated[13] "that over 90 per cent of the power load of the

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* Director, Anthracite Institute Laboratory.

† With acknowledgment to D. L. Corgan, Anthracite Institute, for data on use and performance of gas producers.

[13] Haslam and Russell: Fuels and Their Combustion, 9.

United States is located east of the Rocky Mountains, whereas this area possesses only 30 per cent of the potential water power." Thus even with a complete development of hydropower, vast industrial areas in the east would still be dependent upon combustible fuels for their power.

While it is true that water power has advanced rapidly in recent years, it must be remembered that the development of this energy was negligible until the beginning of the present century, and that there were really no sensational advances in water-power development until after the World War. It was therefore inevitable that the strategically located lower-cost projects should have received attention during a boom period, thus leading to the rapid expansion that Mr. Tenney shows as having occurred between 1921 and 1928.

A second factor of interest in connection with the utilization of coal is the almost unlimited possibilities of modern coal-burning gas producers and producer-engine combinations. In many foreign countries price increases and fuel shortages have resulted in the acceptance of large numbers of these devices; in America their use is comparatively insignificant.

The Duetz Motor Co., in Germany, manufacturing producers and gas engines, claims a production of 1 kw-hr. from each 1.23 to 1.5 lb. of coal, an economy that equals if not excels the most economical steam plants in this country. Their guarantee is placed at 1.35 lb. of coal per kw-hr., an extremely attractive figure in view of the simplicity and proven practicability of their equipment. Other operating charges are almost negligible, one gallon of lubricating oil per 1240 kw-hr. and one man on plants up to a capacity of 3000 kilowatt-hours.

H. N. EAVENSON,* Pittsburgh, Pa. (written discussion).—In regard to the decrease in per capita coal consumption, if five-year averages are taken in order to smooth out the annual irregularities, it will be seen that the decrease in per capita consumption of coal actually began in 1918, and that in 1932, the last year for which the figures are available, the per capita consumption was only about 60 per cent of what it was at the peak. The decrease since 1918, with one slight exception, had been practically steady even while the population curve and the curve of manufactures had been increasing steadily. Too little attention has been paid to this fact, and until the downward trend of this curve is checked and an upturn started, there will not be much of an increase in the amount of coal used.

Another fact to which the author calls attention but does not emphasize is that there is almost certain to be a considerable increase in the cost of coal in the future. The increased wages, which are inevitable under present conditions, will increase the labor cost and therefore promote the use of the competing fuels, which will further increase the cost of coal by decreasing its operating time and adding to the overhead items in the cost of production. This, with the increased costs due to regulation, which apparently are inevitable, is certain to make a considerable increase in the price of coal.

The author's statement about the uneconomic conditions that may retard the future development of hydroelectric plants are, of course, true, but at the same time we see the greatest development of hydroelectric plants fostered by the Government, and paid for by Mr. John Taxpayer, that the country has ever seen, and apparently there is no likelihood of stopping this development; and once the plants are built it is almost certain that they will be operated whether they are economically sound or not.

These plants will inevitably require the construction of stand-by steam plants, but even these will not add to the amount of coal in use, because most of the power that will be furnished from these plants can be already furnished from existing steam plants.

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* Consulting Engineer.

One thing that has always impressed the writer about the power industry is the great weight that is attached to the ash-fusion point of coal in the United States as compared with the little attention that is paid to it in foreign countries. A very careful search of all the literature available recently found only one reference in foreign literature attaching any real importance to this factor, the only reference to it really being in the fact that it causes coal to clinker, and whatever weight is attached to it is due to this fact entirely, and it is very rare in any of the combustion literature to find any reference whatever to ash-fusion temperature.

The same remark applies in some measure to the ash content of coals. In this country we seem to think that the very low ash is desirable, but in some of the recent English literature it is suggested that for combustion purposes coal having as high as 11 per cent ash is better than one having less than that. While this may be an extreme assumption, we seldom see reference in the foreign literature to the very low ash in coals that we are accustomed to in this country.

It is certainly gratifying to see from the author's conclusion that he expects a marked growth in coal consumption for stationary power purposes, and any coal man will join in hoping that this conclusion is sound.

E. H. Tenney.—The reference in Mr. Johnson's discussion to the gas producer combinations is timely. There is a good deal of such equipment being developed and installed and operated, some of it in the United States, and there is a field for it. The development of the internal-combustion engine in the oil fields is a parallel development. The more we can get these people to use electricity and become electrically minded, even in the far regions, down in the extreme Southwest, where a good deal of such equipment is being used, the more extensive will be the use of electricity generally, and, in view of the conclusions that we reached in these studies, if they are accurate, the greater will be the use for coal.

As to the increase in the industrial load brought out by Mr. Tobey, at this particular time we do not know just what to think of it. The very slow increase in the industrial power usage during the past few years was not unexpected, and the continued increase in the domestic load was expected. But during the past six or eight months, especially during the past six months, the rapid increase in the industrial use of power has been so marked that we do not know what to make of it. If that rate of increase continues, there is no doubt that some of us are just going to be swamped with the requirements and the demands for power.

In connection with Mr. Tobey's remarks on the undesirability of low machine load factor, this is very true and probably always has been and, because of the nature of the loads, especially the domestic load, always will be that way. Still, if, in this increase in the industrial use, it can be worked out so that the industries will use these large amounts of off-peak power that are available during the night hours and during periods when the lighting load does not come in so strong, it will increase the station load factor and will tremendously increase the amount of power that is put out.

In the St. Louis district during 1930, the load factor was 48 per cent; in 1935 it was 60 per cent, just a little bit higher than the figures that Mr. Tobey quoted, but very definitely emphasizing the point that he brought out. It is a good suggestion, and if it can be followed out to the advantage of all, and if more use can be made of this off-peak power, the coal interests will benefit.

Relation of Steam-generating Equipment to Preparation, Selection and Burning of Bituminous Coal

By E. G. Bailey,* Member A.I.M.E.

(St. Louis Meeting, October, 1935)

The bituminous coal industry faces a real problem, if it desires to retain the position in the power-generation field to which it is economically entitled. More power is probably produced today for electrical energy, for steam-driven and steam-heating industries and processes, for transportation by rail, water, highway and air, collectively, than ever before, and yet the total bituminous coal production has decreased about 38 per cent, from 575,000,000 tons in 1926 to 357,500,000 tons in 1934.

It is not the intention of the writer to recommend coal as a fuel for automobiles and airplanes, but rather to point to these forms of transportation as being responsible in a very measurable degree for reducing the coal consumed by railroads. Petroleum products may still further reduce consumption of locomotive fuel through the extension of the use of Diesel-driven trains. Oil-burning steam-driven locomotives are replacing coal in some places, where railroads operating high-speed passenger trains are willing, temporarily at least, to pay a premium for the fuel that is low in ash and produces less smoke and practically no cinders.

In the navy and ocean-going passenger ships, oil is so firmly established, and has such definite advantages over coal as formerly fired, in steaming radius, in ease of bunkering, cleanliness, and reduction in firing labor, that very outstanding developments in the handling and burning of coal will be required in order to replace oil. For freighters on the Great Lakes, coal is the logical fuel, but there is great and immediate need of radical improvements in its use, else that market also may be lessened.

It is not within the scope of this paper to discuss in detail bituminous coal for either locomotive or marine use, but, rather, generation of power in stationary plants, consisting of electrical generating stations and that very large group of miscellaneous industrial plants, such as paper, rubber, textile, steel and other industries that use electrical, mechanical, and steam power, derived from either hydro or some form of fuel.

Manuscript received at the office of the Institute Sept. 3, 1935.

* Vice President, The Babcock & Wilcox Co., New York, N. Y.

The three factors that have been most responsible for reducing the consumption of bituminous coal, not given in order of importance, are:

1. Hydroelectric power generation.
2. Petroleum; oil and gas.
3. Improved efficiency in combustion of coal, generation of steam and its use in turbines, etc.

Improved efficiency in steam plants has no doubt been the greatest single factor in reducing coal consumption per unit of power output, for it has been reduced in the better electric stations from more than 3 lb. to 1 lb. of coal per kilowatt-hour. Similar reduction in coal consumption has taken place in practically all industrial plants, though to a lesser degree. However, if these economies in the burning of coal and in the use of steam had not been brought about, hydro and oil would have made greater inroads, proportionately, than they have. In fact, the salvation of the bituminous coal industry lies in the direction of still greater efficiency and cleanliness in the use of its product, coal. The consumption of coal per unit of power must be further decreased, and the facility and cleanliness of its handling and use be increased, to protect it against further inroads by other forms of fuel or power generation.

There are three groups of companies most interested in the problem of generating power from coal: (1) the power-plant owners or users, wherever coal is the normal, or even possible, fuel; (2) the manufacturers of coal-burning and steam-generating equipment; (3) the coal producers.

Power-plant Owner

The power-plant owner, usually upon the advice of his consulting engineer, buys equipment that is considered best adapted to meet the anticipated conditions, with reference to capacity, load factor, kind and price of fuel most economical to use, and the efficiency and reliability he hopes to obtain.

After the plant is built, it devolves upon the owner to properly evaluate the different coals available, and produce steam at the lowest over-all cost, including fuel, maintenance, operating cost, capital charges and efficiency. The most difficult part of this procedure is to properly evaluate fuels when they are dissimilar.

While the heating value is the principal factor involved, the character of the combustible constituents, as well as the amount and nature of the impurities, and the size and size distribution of the coal, may each have greater or less influence upon the efficiency and on the cost of burning the coal and handling the refuse. An effort is being made to work out a basis for comparative evaluation of coals from their analyses, but it is doubtful whether in the end it may not be more confusing and inequitable, either to the seller or buyer, to adopt such method rather than to continue to allow each purchaser to use his own method and discretion.

Those actively working along the line of standardizing evaluation factors should keep in mind that each kind of stoker, and each kind of furnace construction, and each capacity range will call for different factors for many of the constituents of coal. There is also a lack of knowledge regarding coking or agglutinating values, and how to apply them. The ideal coal for a multiple-retort underfeed stoker may not be at all suited to a chain-grate stoker, and vice versa. The variations in the constituents may require entirely different factors to properly evaluate coal for the respective types of stokers, while pulverized coal may be less sensitive to variations in any of the factors except the B.t.u. value, but even its relative value may be somewhat affected by the method of ash removal and other factors of furnace and boiler design.

The surest method of evaluating coals for a given equipment and load conditions is actually to operate with each coal over a sufficient period to get all equipment properly adjusted to the coal and enable the operators to learn the best methods of firing, the proper amount of excess air, the capacity limitations, etc.

Every coal-burning plant that falls short of the best that is now known is a potential prospect for oil-burning equipment. To modernize for oil or gas usually costs less than to revamp the coal-burning equipment. If oil or gas also increases the capacity and efficiency and reduces the labor cost over the obsolete coal-burning equipment, the plant owner is often willing that the over-all power cost may be higher. Many plants have changed from coal, when the power costs were lower, with oil at the price then in effect, but when the price of oil advanced, as it has very generally in the past two years, it is very difficult and often impossible to swing back to stoker firing without excessive replacements and repairs, and perhaps a decrease in capacity.

There is greater need for standardization or general agreement on a method of evaluating fuels of great dissimilarity, such as coal, oil and gas. Such fuels may require changes in furnaces and equipment that, when once made, are nonconvertible, except at great expense. The expected efficiency figures and operating cost are usually supplied by the company offering oil or gas as a fuel to replace coal. New equipment purchased may carry the manufacturers' efficiency and capacity guarantees, but the plant owner should carefully check all expected efficiencies and estimated operating costs.

Sometimes when a change is made from coal to oil or gas, the estimated savings are not fully obtained, and proper allowance is seldom made for added explosion and fire hazard. If the boiler feed-water conditions are just on the border line for coal fuel, there is often a series of boiler outages, caused by tube losses in the boiler or perhaps the water walls, due to the higher rate of heat absorption inherent with gas and oil firing, as well as to the fact that the tubes in the furnace row do not have the benefit of the protective ash coating from coal firing. Such a situation may cause an unexpected and unestimated increase in operating cost, due to more frequent internal cleaning of the boiler and wall tubes or to improvement in methods of feed-water treatment, or, perhaps, capital investment in water-treating equipment.

A condition often encountered in older plants is an increasing load simultaneous with deteriorating fuel-burning equipment, so that special high-grade coal or perhaps oil must be burned to carry the load. One solution is additional boiler and fuel-burning equipment. Another is new equipment only, as the boiler has often outlived two or more sets of stokers or fuel-burning equipment. The latter form of change is very likely to be made if low-priced fuel oil becomes available in an erstwhile coal market. The plant owner should be very thorough in his investigation before making such a change, to make sure that he could not better install coal-burning equipment that is an improvement over the types available when the previous installation was made. He should also be prepared for a reversal of the fuel situation when coal is again unquestionably the more economical fuel.

The Equipment Manufacturer

The manufacturers of fuel-burning and steam-generating equipment are a very important link between the fuel producers and the fuel users. Most of the earlier inventions of stokers, oil burners and coal pulverizers originated with persons outside of the fuel-producing interests. Those inventing such equipment usually turned them over to a foundry or machine shop, and only recently have the coordination and joint development of fuel-burning and steam-generating equipment been undertaken by manufacturers themselves on an extended scale.

The equipment manufacturer has naturally kept close contact with his customer, the power-plant owner, and has endeavored to develop equipment to meet his wishes and requirements for whatever fuel it seemed best or easiest to burn. In localities where low-grade coal prevailed, suitable equipment was developed for burning it with reasonable satisfaction and efficiency. Waste fuels, such as wood refuse, bagasse, coke breeze, petroleum coke, etc., have each caused special equipment to be developed, most of which was poorly, or, perhaps not at all, adapted to any other fuel that it might be desirable to use at some later date.

It should be, and now is, the goal of progressive manufacturers to improve and develop furnace and fuel-burning equipment that will burn as wide a range of fuels as possible with maximum efficiency. If low-grade coal can be burned with the same efficiency as high-grade coal, the purchaser can select the coal having the greatest British thermal unit per unit cost on a delivered price basis, with perhaps a few minor factors, such as ash disposal. This will give the plant owner the greatest range for selecting the most economical fuel, and the lower grade fuels will have a proportionately increased advantage.

Pulverized coal, oil and gas can now be burned in the same furnace without changing burners or even shutting down to change from one fuel to the other. In fact, two or three fuels may be burned simultaneously, the total amount of fuel being limited, of course, by the available air supply. A wider range of coal can be efficiently burned with suitable pulverizing equipment and furnace than with any one type of stoker. This, then, is an approach toward a universal furnace where coal, oil, and gas can be burned under similar conditions.

The present tendency in many parts of the country, and especially along the Atlantic seaboard, where the relative prices of coal and oil make them competitive, is to install new equipment that is adapted to burn either coal or oil, with a minimum of cost and delay for the change over from one fuel to the other. The first impulse of most coal men is to oppose this, but upon careful consideration it is seen that, in the long run, it will work out to the advantage of the bituminous coal industry. The most readily converted boiler unit is one in which pulverized coal or oil may be burned without changing any part of the furnace or even the burner. Whichever fuel is most economical may be purchased on contract, or even spot shipments, and the operation of the unit changed at will; or both fuels may be burned at the same time. Such equipment also has the fundamental advantage of being more efficient than a stoker-operated plant, so that coal will naturally be burned a greater percentage of the time considering the greater difference in efficiency and operating costs that usually exists between oil and stoker firing.

Aside from the stoker and the coal pulverizer and burner, the furnace design has greatly improved the efficiency and smokeless combustion of coal. It is now realized that the proper shape and size of furnace are fully as important as the stoker or burner.

One of the most important furnace developments is that of water-cooled walls. Within a few years, water-cooled furnaces have become firmly established as an economic necessity, not only for pulverized coal but for stoker, oil, and gas firing as well. Water-cooled furnaces have helped the utilization of coal having low ash-fusing temperature more than coals with high fusing temperature; they have helped all bituminous coals more than they have helped oil and gas, and they have been largely instrumental in increasing the capacity, efficiency and reliability of all boiler furnaces burning every kind of fuel; they constitute a very important step in the trend toward the universal furnace and fuel-burning equipment.

The power-plant owner is willing to invest more in fuel-burning and steam-generating equipment in order to realize a reduction in power costs through greater efficiency and correspondingly lower fuel consumption.

COAL PRODUCER

The bituminous coal industry has not been a unit in fully appreciating its problem in a broad way. Some branches of the industry still consider "King Coal" an unimpeachable ruler over the power-generating field. The power field has ceased to be a monarchy, and coal comes up for election in many parts of the country each time a new or rejuvenated fuel-burning equipment is contemplated. Often impeachment proceedings are initiated by "hydro" or "petroleum fuels," without waiting for regular election time, and each contract period is likely to open the nominations to a wider range of coals and other fuels than was considered acceptable before better fuel-burning methods were established.

On the other hand, many coal producers have been very much awake to the situation and have cooperated splendidly with the equipment manufacturer and the user in cleaning, preparing and sizing their product to best meet the needs of their customers. They have learned that uniformity in product, both as to size and quality, is of increased value to the user, especially for many kinds of stokers.

The preparation of coal by sizing and cleaning should be carefully considered at each mine before the necessary heavy expenditure is made. This should cover not only the present but the best possible prediction of the future trend. The writer believes that sometimes this expenditure has been carried beyond its economic limit. The future trend will possibly justify less, rather than more, expenditure in this direction. The domestic house-heating market may be a controlling factor in certain localities, and this should be watched carefully, for it may take some unexpected turns if real research is done in this field.

So far as steam coal is concerned, some types of stokers are very sensitive to both the character or rank and quality of coal. The producer must be guided by his market, but he should remember that every economic trend will be in the direction of getting away from such limitations and toward getting better efficiency out of low-grade, low-cost fuel. Reducing the ash and sulfur, and increasing the B.t.u. value, increases the value of coal, and if the increase in value on the delivered price is enough to pay for the cleaning, with fair accounting methods, the investment for the cleaning plant may be a good one, providing the market remains throughout the period in which the investment is amortized.

The sizing of coal is a more uncertain factor, for it does not increase the value of coal for pulverizing. Stoker developments of the future may be along lines in which size and quality will both be less important than at present.

Much that has been done in the preparation of coal has been with a view to gaining an advantage over coal-producing competitors. When these in turn install similar or even more expensive and elaborate equipment, each finds himself burdened with a heavy capital investment, calling for relatively higher production costs, with, perhaps, oil taking the business from both.

Some persons who have studied the trend of the times believe that the solution lies in the direction of processing coal through distillation or hydrogenation into the products of coke, oil and gas, so as to more readily compete with petroleum products on their own ground. That may be one answer, at some future time when petroleum is scarce and at a much higher price than is likely to prevail for many years. The most effective step for the immediate future is to overcome the handicap and break down the prejudice against coal wherever it is the cheapest fuel on a heat-unit basis.

Coal has practically no market except for its heating value. Even where it is processed into coke for blast-furnace, foundry, or domestic use, or into gas in producers or retorts, the value of these products lies in their heat value in a more usable form at a correspondingly higher cost. The preparation of coal at the mines by sizing and cleaning adds to the cost of the heat units sold, often out of proportion to the added utility of the product.

The basic problem of the coal industry and of the railroads that serve it is to take from the ground the heat units latent in their coal and deliver them into the steam pipe as simply and cheaply as possible. Energy can be transported in coal cars more cheaply than it can in electrical transmission lines or gas or steam pipes, hence the heat units in coal should be delivered as near the point of final power consumption as is economically possible. The coal should be burned in such manner that its complex constituents of moisture, volatile, ash, sulfur, etc., will cause a minimum expense and inconvenience.

Some people will say: "This is all being done now." It is being done as well as anyone knows how to do it, in a relatively few plants. How about the many inefficient hand-fired boilers; the antiquated stokers that are unsuited to the burning of any kind of coal efficiently; the myriad of other stoker plants where the coal is not adapted to the stokers, or, perhaps, the stokers are not suited to any coal that is economically available? Some plants fired with pulverized coal may have low efficiency, due to coarse pulverization or to operating with high excess air caused by the use of refractory furnaces or to inadequate ash removal. It needs only one trip through the industrial sections of our country to observe numerous chimneys that give irrefutable evidence that bituminous coal is being burned improperly and uneconomically in many plants.

The additional difficulties involved with coal revolve about three major factors:

1. Coal is a solid and cannot be pumped through pipes or handled as conveniently as the fluid fuels.

2. The composition of its combustible is more complex and therefore more difficult to burn efficiently and smokelessly than gas or oil.

3. The ash is a serious deterrent to combustion in fuel beds, and a handicap in obtaining high ratings and efficient heat absorption, because of its accumulation in the furnace and on the boiler heating surface.

The bituminous coal industry is or should be more interested in overcoming some of these handicaps in comparison with the ashless fuels, gas and oil, which are simpler in their chemistry and can be readily burned in suspension. The industry is advanced in the questions of mining methods, ventilation, etc., also in the technical analysis and chemistry of coal, but much is yet to be learned of the combustion process and methods of improving it wherever bituminous coal is being or should be burned.

The power-plant owner, especially the public utilities, and the equipment manufacturers have been responsible for the greater part of the development in coal burning and steam generation. Much that they have done applies also to oil and gas. The bituminous coal industry should make a more thorough and open-minded survey of the best methods of burning its own product, and the operators should cooperate fully with each other and with the equipment manufacturer and the power-plant owner in still further improving combustion methods. It is believed at the present that the best outlook is in the direction of a universal furnace that will be able to burn various coals and perhaps other fuels. It may be that some improved method of burning coal will be so very good and satisfactory that other fuels will lose their present advantage in efficiency, cleanliness and facility of handling.

Coal producers are too prone to compete against other producers in their own or other coal-mining districts. If one has a high-grade coal that works better on one kind of stoker than on another, he naturally favors that kind of stoker and opposes other methods of burning coal. He goes to extra expense in preparing his coal, all of which adds to the price he expects to receive, while if he were to cooperate with other producers in the bituminous coal industry in the development of a better method of burning any kind of coal, they might better meet the competition of other fuels or sources of power.

The power plants at coal mines offer excellent opportunities for the producer to install modern or even experimental equipment to advance the art of burning coal and to which he could take his customers, there to learn how their plants could be improved. Such opportunities are seldom recognized or utilized. Prospective customers of coal-burning equipment often travel great distances to see some new type of equipment in operation, perhaps burning a coal that is different from the coal they will burn. Sometimes they even ship to other plants, at great expense, some of the coal that normally they burn or wish to burn, to make sure that it is suitable to the equipment they contemplate purchasing. How much simpler and how much more effective in his sales effort it would be if the coal producer could demonstrate his own coal at his own mine power plant on the most modern burning equipment best suited to it.

Some of the more progressive leaders in the bituminous coal industry have realized the need for research, and some steps have been taken in the form of committees in this Institute as well as active work sponsored by some branches of the industry. The problem justifies much more thorough study into the best scope and magnitude of activities than seems to have been given it as yet. The manner in which the gas and petroleum industries have approached their problems should be investigated and perhaps followed in certain respects.

Research is a much used word in recent years, and many people and institutions think they are doing research work when they spend some money in library, laboratory and leisurely thinking, which do not bring in immediate financial returns—and very often no returns at all. Too many people blindly follow the spirit of the times and brag about how much money they spend on research, and pay too little attention to the tangible returns they receive from it. The definition for research is "diligent search after facts." Too often the search is not diligent, all the facts are not determined, and those determined are not promptly put to remunerative use.

The bituminous coal industry should have many more statistics than it now has regarding the actual consumption of its product to different industries, uses and types and efficiency of burning equipment. The operators should be fully cognizant of the trends toward other kinds or quality of bituminous coal and especially toward or from other fuels. Statistics on sales of coal-burning and oil-burning equipment are not sufficient. They do not go far enough to give any accurate picture of the coal tonnage involved.

Some branches of the coal industry have considered it advisable to develop or sponsor certain coal-burning equipment. This has sometimes been started with only a meager knowledge of what had actually been done along similar or competitive lines. Great caution should be exercised before going too far in this direction. There are a great many factors involved. Conditions change and new developments take place so quickly that the coal industry should be free to follow and take advantage of new developments, and not be tied to an investment or a fixed policy that would be very expensive to change and perhaps suicidal not to change.

The bituminous coal industry, including the coal-carrying railroads, has been very backward in cooperating with its customer, the power-plant owner, and still more unacquainted with the equipment manufacturer.

The most constructive policy the industry can follow is to minimize competition within itself, both as to individual companies and different districts, and face the common problem with the equipment manufacturer and the power-plant owner for a better solution. All bituminous coal has enough volatile, tar and smoke-producing matter, and enough ash, so that the fundamental problems that are common to all coal should be worked out collectively, instead of individual producers striving for a false advantage because of a difference in the volatile, ash, or its fusing temperature, of their coal over a competitor's. This individual inter-industry competition has often misled the customer and the equipment manufacturer into thinking that it was only necessary to go part way in the problem of coal-burning development to meet the requirements of a certain coal specially prepared, later to find that some other fuel replaced coal altogether.

The ultimate goal should be a universal furnace in which any coal, of whatever rank or grade, can be so efficiently burned that other fuels will be able to compete only when their price and B.t.u. value justifies their use.

DISCUSSION

(*C. J. Sandoe presiding*)

E. H. Tenney,* St. Louis, Mo.—I agree with a great many of the points Mr. Bailey has brought out. It is certainly a fact that the fuel-producing people must cooperate with the equipment manufacturers and the people that burn the fuel. There has been a lack of cooperation between the fuel-producing people and the fuel consumers, especially in the central station line. That probably has not been intentional. It has been due merely to the fact that there has not been a realization of its importance. Often the fuel people have inadvertently supplied fuels that were not at all adaptable to the particular conditions that had to be met. Mr. Bailey has brought out that point excellently in his paper.

J. Griffen,† Pittsburgh, Pa.—I agree with Mr. Bailey when he says: "The basic problem of the coal industry and of the railroads that serve it is to take from the ground the heat units latent in their coal and deliver them into the steam pipe as simply and cheaply as possible." I do not altogether see eye to eye with him when he raises the question as to mechanical preparation fitting into that picture. Mechanical preparation does absolutely fit into that picture because it usually offers an opportunity to actually mine the coal much more economically than it has been mined. In other words, mechanized mining produces cheaper coal at the pit mouth but usually with an increase in impurity content, which really should not be shipped to the market. Mechanical preparation enables this cheaper but dirtier mechanically mined coal to be prepared to a high standard of quality, so that the cost of heating value delivered to the consumer is lowered.

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* Union Electric Light and Power Co.

† Koppers-Rheolaveur Co.

S. P. Burke,* Morgantown, W. Va. (written discussion).—Certainly all who read Mr. Bailey's paper will appreciate the timely and lucid discussion he gives concerning present trends and probable future developments in the utilization of coal for steam-raising purposes. His long and competent experience in this field gives great weight to his opinion. It is worthy of note, however, that the history of technological development in other industries is somewhat at variance with Mr. Bailey's predictions. The chemical industry is an outstanding example. Here we find continuous improvement in the refinement of the producer's product to meet consumer specifications that have steadily grown more exacting. The preparation and refinement of products is invariably conducted by the producer, not by the consumer. In the fields of competing fuels, we find essentially a similar trend in the oil and gas industry. Advancing technology requires improved performance of the equipment for utilization, which in turn demands steadily decreasing tolerances in the variations of the fuel supply.

It is interesting to observe that the effect of these trends in the industries referred to (particularly the gas industry), has made necessary a closer and closer liaison between the manufacturer of the utilization equipment and the producer of the fuel. To a considerable extent, this has affected the entire business organization of the industry. To be sure, increased knowledge of the process of combustion will permit the development of coal-burning equipment that can utilize coals of wider variation in properties. On the other hand, the more exacting demands on the part of the consumer may more than keep pace with this and make it necessary for the producer to prepare his product to meet sharper and sharper specifications. At any rate, this appears to be the history of other industries in which technological development has been more rapid than in the coal industry.

In reading the discussion of the transfer of energy (p. 220), I received the impression that the author believes that the freight car is the most economical method of energy transportation. This is not true, since on a B.t.u. basis oil can be transported more cheaply, and also natural gas, over a distance equivalent to a "short haul." On a basis of British thermal units utilized, the transportation of oil and gas shows to better advantage.

H. N. Eavenson,† Pittsburgh, Pa. (written discussion).—The author's presentation of the subject is so complete that it rather precludes discussion. In view of the attempts to evaluate the various factors affecting combustion within the last few years—in which I have been much interested—it is refreshing to find the author calling attention to the fact that the only sure way to do such evaluating is by actual operation of the coal in question in the equipment under consideration. Most attempts at such evaluation have covered too many kinds of apparatus over too large an area to reach results in which much confidence can be placed. Any such attempts must take into consideration the equipment in which the coal is to be used, and in any market area the prices of two coals will vary, largely depending upon their use.

The author's statement that the preparation of coal often increases the cost out of proportion to the increase in utility is true, and very timely. There is a great deal of fashion in the coal industry, and many operators are forced into some kinds of preparation not needed for their coal—and often not suitable for it—because competitors have prepared their product in certain ways. Much of this trouble is due to poor salesmanship, in not convincing the user of the real value of the product instead of its form.

Some real comparative tests of the costs of burning various fuels, including liquid fuels, are being made by Battelle Memorial Institute, and while these tests are on domestic furnaces only, they will undoubtedly give some light about industrial uses as well. It is expected that this information will be available in the latter part of 1936.

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† Consulting Engineer.

The author's statements about the lack of cooperation in the industry in research and other directions are only too true, and we can only hope that a better spirit will soon prevail and will enable the goal described in the closing sentence of his paper to be reached.

E. Steidle,* State College, Pa.—We will agree with Mr. Bailey that the burner must be made to fit the needs and that the future of coal depends on careful planning. It is significant that after all these years we do not actually know what coal is and are limited in our knowledge of the chemistry and physics of combustion. Coal will continue to mean power but we must comprehend and take into account the political, social, and economic consequences of all new developments in power. In this respect, it is possible that we will regret some day, when it is too late, that we have burned oil. While somewhat the same logic holds for bituminous coal, still we do have by-products now and the position is not so pronounced, owing to larger reserves. I am told that a Diesel engine can be operated on practically everything that is combustible; it is possible that vegetable oil in another half century will be considered a bigger competitor of coal than petroleum is today. Again, the problem is made more complex with the possibility, remote as it may seem at the present time, of burning de-ashed powdered coal in the Diesel engine. Last but not least, we may expect commercial fuel oil from hydrogenated coal within the next decade, which would provide a new market for bituminous coal in this country.

W. G. Christy,† Jersey City, N.J. (written discussion).—I heartily agree with almost everything Mr. Bailey says. It has always seemed to me that there is less cooperation among representatives of the bituminous coal industry than in any other business of which I have knowledge. In one large city I found that new customers were given lower prices, on the same kind and quantity of coal, than consumers the coal companies had been serving for years. The soft-coal industry has often given only half-hearted support to smoke-abatement campaigns. It would seem that men in this line of business would want to cooperate with a smoke-abatement educational program and teach their customers how to burn coal smokelessly and efficiently. In many instances, producers and distributors of bituminous coal have actually worked against any concerted move to improve smoke conditions. This was, apparently, done under the mistaken idea that smoke abatement would hurt their business.

Many sales of stokers and other modern coal-burning equipment have been killed, or postponed, by coal men. They have informed their customers that the estimated savings were greatly exaggerated by equipment salesmen and the investment would not be justified. The writer has always felt that this was a shortsighted policy. As Mr. Bailey points out, improvement in the method and efficiency of burning coal is the best weapon the coal industry has to combat other fuels.

The ignorance of many representatives of the coal companies about their own product is appalling. Many can not even give an accurate proximate analysis of the coal they are attempting to sell. If asked to tell the characteristics of their products, or give correct instructions in burning it, or recommend the right kind of coal for certain equipment, they are at a total loss.

This industry has been backward in carrying on research work in the use of bituminous coal. Only within the past two or three years has comprehensive research work been started by the industry. Previously, such activities were carried on by the U.S. Government, users, educational institutions and research laboratories.

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† Smoke Abatement Engineer, Department of Smoke Regulation, Hudson County, New Jersey.

The loss of coal tonnage in the past few years is partly attributable to some of the short-sighted policies outlined above. If we compare these policies with those of competitive fuel companies and with those of other industries, the differences are startling. In comparison with liquid and gaseous fuels, coal is handicapped by its ash content. From the standpoint of the public, we must give serious thought to the question of air pollution. In this discussion we will omit any argument about smoke, on the assumption that any fuel can be and is being burned smokelessly. There remains the problem of cinders and fly-ash from combustion of coal. The time has now arrived when producers, distributors and users of coal must give their best attention to this problem. The public is demanding that immediate steps be taken to greatly reduce, if not eliminate, pollution of our atmosphere with cinders and fly-ash.

E. G. Bailey (written discussion).—I am glad Mr. Griffen brought out the point he did, because I did not intend to say anything against cleaning, providing it paid for itself. His remarks intimate that by complete mechanization within the mine they can save enough to pay part of the cleaning costs, and the better market conditions will pay for the rest. The real problem is to see that through salesmanship or fad the producers are not led into an expenditure that is in the last analysis unjustified.

Professor Burke's comments are very much appreciated, but where a consumer needs a chemical he prepares specifications according to his requirements. However, the coal producer must remember that he is not selling coal; he is selling B.t.u., and, so far as I know, the specifications for B.t.u. have not yet changed. The cost per B.t.u. in the rough is not always on an equal basis for comparison, and the coal producers have been laboring under difficulties because they have not cooperated with the equipment manufacturer and the consumer in getting the best results under the handicaps of ash, smoke and agglutinating properties.

The question of transporting fuels more cheaply by other than rail is a question for individual conditions. Certainly one would not expect natural gas to be hauled by rail, and I am afraid it will be some time before coal will be transported through pipes for distances greater than a few thousand feet.

My paper related primarily to coal, and I did not intend that my remarks relating to coal should be confusing to anyone thinking of fuels, particularly in the matter of transportation.

Variables in Coal Sampling

By J. B. Morrow,* Member A.I.M.E., and C. P. Proctor,† Junior Member A.I.M.E.

(New York Meeting February, 1935)

With numerous plans under consideration for coal classification, and with the advent of the Bituminous Coal Code, the intelligent sampling of coal has become increasingly important. To us it is rather significant that there have been very few fundamental studies of coal sampling in American technical literature since those of E. G. Bailey, which appeared in the *Journal of Industrial and Engineering Chemistry* in March, 1909. He covered the subject up to that time so thoroughly and in such an excellent manner that perhaps many other later students have felt that they could add very little to his findings. The recent studies of the British Engineering Standards Association have indicated the possibility of a somewhat different interpretation of Bailey's results, permitting the taking of smaller samples, and perhaps it will now be helpful to reopen this question.

C. W. H. Holmes, in an informative series of articles on coal sampling recently published in *Colliery Engineering*, makes two rather interesting statements, as follows:

> It is scarcely surprising to find a lack of interest and knowledge regarding the sampling of coal not only in the mining industry itself but also among consumers of industrial fuel: at its best coal sampling is a dirty business, regarded as a nuisance and an expense by the commercial man, and usually as a nuisance and nothing else by the chemist.
>
> The writer can conceive only two classes of people who have never experienced any difficulty in obtaining representative samples of coal from wagons; those who have never tried to, and those who have never had the necessity or desire to check the accuracy of their work.

The primary purpose of this paper is to present some tabulated data of the more significant results of a sampling investigation undertaken during the last four years to obtain certain fundamental data on the variables connected with coal sampling and to point out that often assumptions have been made that are not in accordance with the facts. One of the standard sampling methods states:

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> The chance inclusion or exclusion of 10 lb. too much or too little of impurities in or from an otherwise representative sample of 100 lb. would cause the analysis to show an error of approximately 10 in ash percentage, and in heat units of approximately 10 per cent, whereas for a 1000-lb. sample the error would be approximately one-tenth as large, being the same whether the sample is collected from a 1-ton lot or from a lot consisting of several hundred tons.

This assumes that the ash in the impurity is 100 per cent, which is not a fact. If we assume 1.60 sp. gr. as a dividing line between coal and impurity, 50 to 65 per cent ash would be a more accurate figure, and if we assumed 1.40 sp. gr. as the dividing line the average ash in the impurity would probably run from 35 to 50 per cent ash. Obviously a substitution of more accurate values would result in a reduction of the size of the gross sample necessary to give the same accuracy as that mentioned in the standard under discussion.

Factors Governing Variability

The average quality of a particular coal is important and is fairly easy to determine by averaging a number of samples over a period of time; providing, of course, that there is not a constant error in the sampling procedure.

But variability as well as the average quality is important. We mean by variability the distribution of values from the average. A uniform coal will have a low degree of variability. It is generally accepted that the results of a series of samples will illustrate the degree of variation of the coal itself. Rather the results should be considered as seeming variations due in part to the true variation of the coal and in part to the sampling and analysis. These variations depend on the following factors, which must be considered in any study that undertakes to reveal the true quality and variability of a given coal:

1. The true variability of the coal itself.
2. The accuracy of taking the gross sample.
3. The accuracy of reduction of the gross sample to minus 4 mesh.
4. The accuracy of the reduction of the minus 4 mesh sample to minus 60 mesh.
5. The accuracy of the analysis of the sample.

In many cases we have found that the last four factors may cause more variation in the results than that caused by the true variability of the coal itself.

In general, regardless of results that may be obtained, there is really very little variation from one day to another in the quality of the mine-run produced from any one mine. Any changes that occur are gradual and can be detected only by continuous sampling over a long period, unless there is a change in mining methods or in outside preparation.

The foregoing statements, however, are not applicable to any one car or group of cars of one size or of mixed sizes, particularly if these samples are taken at one certain period during the day's loading.

It seems to us that many of the misconceptions on coal quality are due to erroneous interpretations of the analytical results. There still exists a very common idea that the results of any one sample will represent the true quality of the coal sampled. The common practice of reporting ash to the nearest one-hundredth of one per cent and other factors with similar seeming degrees of accuracy have led many people to believe that these hair-splitting figures have some real meaning, but, as a matter of fact, the results from one sample are often only one piece of a jig-saw puzzle and do not represent the whole picture.

It is obvious that for commercial application it is economically impossible to expect the same accuracy of sampling on a material of low value, such as coal, as compared to that which commercially may be necessary on a product such as a high-grade ore.

In custom smelter practice it is common to take samples representing 5 to 10 per cent, or even more, of the total tonnage of any one consignment, and to send this entire sample to the sample mill for reduction; the rejects, of course, are sent back to be mixed with the furnace charge. Obviously, it is impossible to adopt a similar method with coal, for one important reason: the crushing to minus ¼ in. generally greatly reduces its value, particularly in steam and domestic markets, which is not true in ore sampling.

It would seem that there should be a distinction between the sampling of coal for research and for commercial purposes. Commercial sampling, at least, should recognize that the odds are against any one sample representing accurately the true quality of the coal sampled. When this fact is finally appreciated and the need for a tolerance conceded we shall have taken great strides toward the goal of proper coal sampling and proper interpretation of analytical results.

We must admit that coal is a heterogeneous material, and we would be better off if we followed the lead of other industries and expressed the results by statistical methods instead of attaching a high degree of accuracy, which does not exist, to individual results.

The size-weight theory of coal sampling as introduced by Bailey, and the later studies of E. S. Grumell, A. C. Dunningham and T. W. Guy, all recognize that there is, almost always, a greater variability in high-ash coals. Grumell and Dunningham recommend increasing the number of increments and thereby the size of the sample if the reported results are to fall within the same limits for both high-ash and low-ash coals. To determine variability it would seem that on the same size of coal we should always take the same size of sample, consisting of the same number and same weight of increments; then the range in the reported figures would be a more reliable guide of variability. These figures, however, would include the errors of sampling, reduction and analysis.

With these factors in mind there appears to be a definite need for usable and generally accepted standards for the commercial sampling of coal. These standards should recognize the limits of accuracy in the determination of the average quality and the degree of variability. It is equally important that the cost of taking and preparing the sample should bear some relation to the value of the product.

Investigators have recognized that an expression of the limits of accuracy is necessary in coal sampling. Bailey recommended that the accuracy be such that the maximum error encountered once in 10,000 times would be within plus or minus 1 per cent from the true average.

The British Engineering Standards Association recommends an accuracy such that 99 per cent of the samples would be within ±1 per cent from the true average.

To express variability, the usual practice is to show the percentage of samples that fall within various limits from the average. This may be expressed by a tabulation or by graphical methods.

Bailey and the British Engineering Standards Association have shown that the distribution of sampling results follows closely the laws of probability. As the distribution curve of probabilities is fixed by the value of the probable error, it is possible to express the distribution in one figure, eliminating the need for tables and graphs.

By definition the probable error is the value of the limit from the average within which 50 per cent of the results will fall. The probable error (*r*) signifies the following distribution:

LIMITS	PERCENTAGE OF RESULTS
Average ± *r*	50.0
Average ± 2*r*	82.3
Average ± 3*r*	95.7
Average ± 4*r*	99.3
Average ± 5.7*r* (maximum error)	Expected once in 10,000 results

For instance, if the average ash is 8.0 per cent and the probable error is 0.5, the following distribution is to be expected:

LIMITS	PERCENTAGE OF TESTS
Average ± *r* equals 8.0% ± 0.5 or 7.5 to 8.5%	50.0
Average ± 2*r* equals 8.0% ± 1.0 or 7.0 to 9.0%	82.3
Average ± 3*r* equals 8.0% ± 1.5 or 6.5 to 9.5%	95.7
Average ± 4*r* equals 8.0% ± 2.0 or 6.0 to 10.0%	99.3
Average ± 5.7*r* equals 8.0% ± 2.9 or 5.1 to 10.9%	Expected once in 10,000 tests

The actual distribution usually checks closely with the probable distribution, with slight exceptions that will be noted later. In general, the actual distribution of a large number of samples will check the probable distribution more closely than a small number of samples. This would indicate that when 100 samples are taken, for example, and the probable error calculated, the probable distribution will more nearly represent the true distribution than will the results from the 100 samples.

G. B. Gould, in his book, "Steam Generation Steps Ahead," shows the variation in results of 2246 samples of Appalachian bituminous coals; 23 coals are represented in the group under 10 per cent ash and 7 coals in the group over 10 per cent ash.

We have calculated the probable error and arrive at the following standard distribution which checks extremely closely with his actual figures.

TABLE 1.—*Appalachian Bituminous Coals, Variation from Average*

Limits	Under 10 Per Cent Ash		Over 10 Per Cent Ash	
	Actual	Theoretical	Actual	Theoretical
± 0.5	32.0	31.5	25.0	25.5
± 1.0	57.0	57.5	48.0	49.0
± 1.5	77.0	78.0	69.0	67.5
± 2.0	87.0	89.5	80.0	81.5
± 2.5	92.0	95.5	85.0	90.0
± 3.0	96.0	98.5	92.0	95.0
± 4.0	98.0	99.8	95.0	99.2
± 6.0	100.0	100.0	100.0	99.5
±10.0				100.0
Probable error................	0.84		1.03	

It is not within the scope of this paper to discuss the mathematics of the probable error. The probable error r may be found by the following simplified Peter's formula as recommended by the British Engineering Standards Association:

$$r \text{ equals } 0.85 \times \frac{\Sigma p}{n}$$

r equals the probable error

where

Σp equals sum of the deviations from the average

n equals number of tests

VARIATIONS IN COMMERCIAL SAMPLING

Observed Variation.—The tables in the appendix are the results of sampling some 7000 cars of coal. We present these data to point out the variations that actually were obtained in the commercial sampling of coal following the idea of increment sampling as suggested by the British Engineering Standards Association. It must be borne in mind that these variations are total variations and include those caused by the true variation of the coal itself and variations caused by errors in: (1) obtaining the gross sample, (2) the preparation of the gross sample, (3) the analytical errors.

We would emphasize the fact that the actual values as shown are not necessarily applicable to all coal sampling but we feel that the principles illustrated by these data can be applied in a study of any coal. It is hoped these results will encourage further studies of a similar nature.

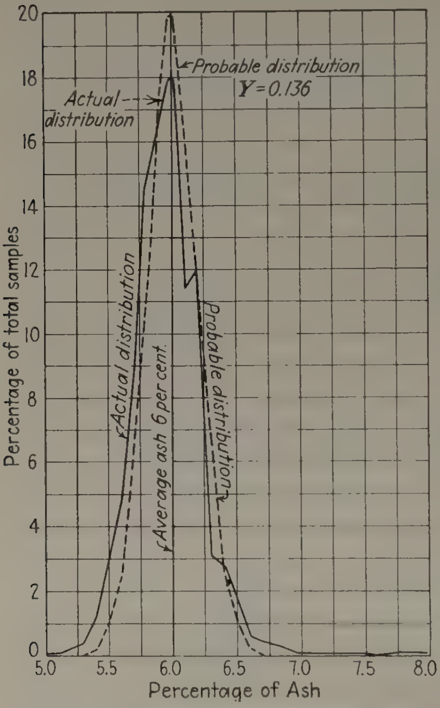

FIG. 1.—COMPARISON OF ACTUAL WITH CALCULATED PROBABLE DISTRIBUTION AT EACH ASH PERCENTAGE OF MECHANICALLY CLEANED 0 TO 4-INCH COAL. ASH DISTRIBUTION OF 2410 INDIVIDUAL CAR CLEANED.

Inspection of Data.—As previously mentioned, Bailey and British Engineering Standards Association *Bulletin* No. 403 point out that the sampling results on coal follow the laws of probability. Other writers have investigated the application of the laws of probability to the sampling of other materials.

We attempted to plot the data from our tests but found, as others have, that 100 results per test were insufficient to give a smooth curve if the individual data were used. The Manual on Presentation of Data (1933) of the American Society for Testing Materials states that "in the opinion of the Committee, the true shape of the curve is not determined unless the number of samples is greater than 250."

At one plant we sampled 2410 individual cars, each containing the same size of mechanically cleaned coal. A graph was made of these data (Fig. 1). The agreement between the actual and probable distribution is very close except for a few high-ash samples.

Variation of Actual Results from True Probability.—Let us assume that we have a coal of 5.0 per cent ash and 1.00 per cent sulfur. It is possible that occasionally a sample would run above 10.0 per cent ash or above 2.00 per cent sulfur, but it would not be possible for a sample to run less than zero per cent ash or sulfur. Thus it is possible to obtain samples that deviate in ash or sulfur more on the plus side of the average than on the minus side.

A small amount of material of low fusion point added to a medium or high fusion point coal will usually have more effect on changing the resulting fusion than the addition of equal amounts of high-fusion material. The addition of lower fusion point material to low-fusion coals may change the fusion only slightly but a relatively large amount of higher fusion material must be added to materially increase the fusion temperature. Thus, usually, the largest variation from the average ash or sulfur should be on the plus side, but on the minus side for fusion. Our results usually show that these conditions actually exist.

The results of determining the percentage of samples above the average is tabulated in Table 2.

TABLE 2.—*Percentage of Samples Above Average*

	Ash	Sulfur	Fusion
Percentage of results above average	48	47	52
Number of tests	43	43	43
Number of units sampled per test	100	100	100
Percentage of tests containing maximum deviation on the positive side of average	74	92	33

This deviation from the true probability curve is shown by the second approximation on page 24 of the A.S.T.M. Manual on Presentation of Data, 1933. The right-hand curve would represent ash and sulfur results and the left-hand curve would represent fusion results.

CAUSES OF VARIATION IN GROSS SAMPLES

The Coal Seam

Underground coal varies in the seam both horizontally and vertically. The vertical variation is generally the major factor in any one mine; the horizontal variation is not so often important in one mine although different districts may show wide variation, especially in the sulfur content. Fig. 2 shows the vertical variation as it actually exists in one mine. There is often a variation in the size percentage and therefore in the ash content of the various sizes produced from the same mine, depending on whether the coal comes from entries, rooms, or pillars. It is particularly noticeable in old pillars that have been standing under relatively heavy pressures. The resultant crushing causes more impurities to go into the slack and thereby increases its ash content. This is one of the reasons that makes it advisable to sample a full day's output of the mine to determine the average. It also indicates the importance of relying on averages rather than on individual results.

Loaded Product

Tables 3 to 9 show the variations in loaded product by individual pieces, by increments, and by specific gravity fractions. Most of the testing was done with 2 to 4-in. coal because this was the easiest size to handle. The range of ash of the smaller sizes shows that they contain pieces from all parts of the seam, whereas the plus 6 in. and 4 by 6-in. sizes consist largely of coal from the low-ash bench above the bearing-in bands. The 4 to 6-in. size is the most uniform of the five sizes; i.e.,

		Raw Coal, Per Cent	1.60 Specific Gravity Float, Per Cent	Height, In.
	Ash,	9.8	9.1	10
	Sulfur,	2.20	1.91	
	Ash,	11.2	10.7	9½
	Sulfur,	1.30	1.25	
	Ash,	10.3	9.9	9½
	Sulfur,	1.20	1.06	
	Ash,	5.6	5.2	15¼
	Sulfur,	0.95	0.92	
	Ash,	5.3	4.5	15
	Sulfur,	0.85	0.81	
Coal and Bands	Ash,	30.8	12.3	½ 2¾ ¾
	Sulfur,	0.85	1.00	
	Ash,	6.4	5.4	23½
	Sulfur,	1.20	0.97	
	Ash,	16.7	12.0	6
	Sulfur,	2.85	2.35	
Average for Seam, Excluding Bands and Coal Between	Ash,	8.2	7.1	
	Sulfur,	1.37	1.16	

Over-all Height 92¾ in.

Fig. 2.—Vertical variation in a coal seam.

it has the lowest probable error of individual pieces. Here again we must emphasize that these results include all of the variables due to sampling and analysis as well as those due to actual differences in quality. These results show clearly that coal is a heterogeneous substance; this applies to individual pieces, to single increments or to specific gravity fractions. We call particular attention to the variation in fusion temperature of ash of individual pieces of coal.

It is often thought that fine coal is the same whether it results from crushing coal or is the natural fine coal that is produced in mining. In Table 10 are given some analyses of 1 to 2 in. and minus 200-mesh fractions of run of mine coal.

TABLE 3.—*Variability of Percentage of Ash of* 100 *Pieces of Different Sizes of Cleaned Coal*

Individual pieces selected at random from flow of coal at regular intervals, mine A.

	Percentage of Pieces				
	Plus 6 In.	4 to 6 In.	2 to 4 In.	1 to 2 In.	½ to 1 In.
Number of pieces	100	100	100	100	100
Ash: average per cent	4.9	4.7	6.7	8.0	9.3
Maximum per cent	12.7	11.0	42.1	55.2	65.9
Minimum per cent	2.1	2.4	2.0	1.7	1.6
Limits of per cent:					
Less than 2.0				4	2
2.0– 2.9	16	6	9	8	8
3.0– 3.9	18	37	27	13	13
4.0– 4.9	23	27	13	13	19
5.0– 5.9	21	10	9	8	10
6.0– 6.9	10	9	9	10	7
7.0– 7.9	5	4	9	14	7
8.0– 8.9	1	3	4	9	5
9.0– 9.9	1	1	3	4	6
10.0–14.9	5	3	12	9	9
15.0–24.9			3	3	3
25.0–44.9			2	4	9
Over 45.0				1	2
Deviation from average per cent:					
± 0.5	26	20	6	15	6
± 1.0	46	52	20	23	12
± 1.5	61	76	27	31	15
± 2.0	75	85	33	38	17
± 2.5	91	89	41	43	23
± 5.0	95	97	91	78	54
± 10.0	100	100	96	94	87
Over 10.0			100	100	100
Probable error	1.25	1.11	2.94	3.81	5.54

TABLE 4.—*Variability of Percentage of Ash of Individual Pieces of Cleaned Coal*

Individual pieces of 2 to 4-in. and 1 to 2-in. cleaned coal taken without observation at regular intervals.

	1 to 2-in. Cleaned Coal								2 to 4-in. Cleaned Coal							
Mine...	A		B		C		D		A		B		C		D	
Test	1	2	1	2	1	2	1	2	1	2	1	2	1	2	1	2
No. of Pieces	200	200	200	200	200	200	200	200	100	100	100	100	100	100	100	100
Ash: average per cent	6.5	6.6	7.0	6.6	5.2	5.2	6.2	6.5	6.2	6.1	5.2	5.6	4.7	4.4	6.2	7.0
Maximum per cent	43.5	25.6	87.0	75.1	24.6	33.3	16.6	17.9	42.2	32.8	50.1	28.2	23.6	15.6	22.2	24.4
Minimum per cent	1.8	1.5	1.5	1.4	1.7	1.6	2.0	1.9	1.8	1.5	1.4	1.7	1.7	1.6	2.0	2.9
Limits of per cent:	PERCENTAGE OF SAMPLES															
Less than 5.0	49.5	41.5	54.0	58.0	65.0	60.5	45.0	46.0	54.0	51.0	69.0	60.0	74.0	70.0	47.0	41.0
5.0– 9.9	36.5	41.5	34.5	30.5	28.0	35.5	42.5	37.0	33.0	37.0	24.0	26.0	21.0	26.0	37.0	43.0
10.0–14.9	9.5	11.5	5.0	5.0	3.0	2.5	11.5	13.0	7.0	7.0	5.0	9.0	2.0	3.0	12.0	8.0
15.0–19.9	2.0	4.5	2.0	2.5	3.0	0.0	1.0	4.0	3.0	3.0	0.0	3.0	1.0	1.0	3.0	6.0
20.0–24.5	1.0	0.5	1.5	0.5	1.0	0.5	0.0	0.0	1.0	1.0	0.0	1.0	2.0	0.0	1.0	2.0
Over 25.0	1.5	0.5	3.0	3.5	0.0	1.0	0.0	0.0	2.0	1.0	2.0	1.0	0.0	0.0	0.0	0.0
Probable error	2.81	2.52	3.59	3.58	2.08	1.79	2.05	2.44	2.80	2.81	2.26	2.50	1.82	1.47	2.32	2.60

TABLE 5.—*Variability in Percentage of Ash and Ash Softening Temperature of* 100 *Individual Pieces of* 2 *to* 4*-in. Cleaned Coal*

Individual pieces were taken blindly every three minutes over five hours of operation, mine A.

Number of pieces tested	100	Number of pieces tested	100
Ash: average per cent	5.7	Ash softening temperature: average	2510
Maximum per cent	34.6	Maximum	2700+
Minimum per cent	1.8	Minimum	2150
	Percentage of Pieces		Percentage of Pieces
Limits of per cent:		Limits	
1.0– 1.9	1	2150–2195	2
2.0– 2.9	15	2200–2245	3
3.0– 3.9	28	2250–2295	1
4.0– 4.9	15	2300–2345	5
5.0– 5.9	15	2350–2395	15
6.0– 6.9	7	2400–2445	10
7.0– 7.9	6	2450–2495	9
8.0– 8.9	0	2500–2545	10
9.0– 9.9	3	2550–2595	10
10.0–10.9	2	2600–2645	9
11.0–11.9	2	2650–2695	6
12.0–12.9	2	2700 and over	20
Above 13.0	4		
Deviation from average percentage:		Deviation from average percentage	
± 1.0	28	± 50	22
± 2.0	52	± 100	39
± 3.0	78	± 150	65
± 4.0	90	± 200	93
± 5.0	91	± 200 and over	100
± 6.0	94		
± 7.0	96		
± 8.0	100		
Probable error	2.28	Probable error	103.9

TABLE 6.—*Variability of Percentage of Ash of Specific Gravity Fractions of Individual Sample Increments of 2 to 4-in. Cleaned Coal*

Individual increments taken every 15 minutes for two days during loading.

Mine	1	A					B				C				D			
Test	1	1	2	3	4	5	1	2	3	4	1	2	3	4	1	2	3	4
Weight of each increment, lb.	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25
Number of increments per test	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50
Head Sample																		
Ash: average per cent	6.8	6.2	6.1	6.1	6.3	6.3	5.6	6.0	5.8	5.4	5.1	5.0	4.9	4.9	6.3	6.4	6.3	6.2
Maximum per cent	9.2	8.6	9.2	8.3	9.7	8.8	7.9	11.3	11.2	9.9	6.2	7.0	6.1	6.8	8.2	7.8	7.8	7.6
Minimum per cent	4.7	4.5	4.5	4.0	4.4	4.5	3.3	4.2	3.7	3.7	4.1	3.8	4.0	3.9	5.4	4.9	5.1	5.2
Probable error	0.70	0.58	0.71	0.69	0.83	0.79	0.68	0.84	0.76	0.59	0.39	0.46	0.36	0.30	0.41	0.40	0.40	0.43
1.40 Sp. Gr. Float																		
Ash: average per cent		5.8	5.6	5.4			4.5	4.7			4.7	4.6			6.0	6.2		
Maximum per cent		[illegible].5	8.5	7.5			6.5	6.5			5.7	5.4			7.7	7.8		
Minimum per cent		4.5	4.5	4.0			3.3	3.9			4.1	3.7			5.1	4.9		
Probable error		0.51	0.56	0.53			0.29	0.40			0.21	0.26			0.39	0.34		
1.40 to 1.60 Sp. Gr.:[a]																		
Ash: average per cent		26.0	21.0	23.5			19.6	21.9			22.5	23.1			20.5	20.3		
Maximum per cent		38.0	31.0	32.6			34.3	31.9			36.3	34.0			29.4	25.0		
Minimum per cent		18.5	13.5	14.1			14.6	13.2			16.0	15.0			16.5	15.8		
Probable error		3.0	2.8	3.8			3.9	2.7			3.4	2.5			2.3	1.2		
1.60 Sp. Gr. Float:																		
Ash: average per cent	6.4	6.2	6.1	5.9	6.1	6.1	4.8	5.2	4.9	4.8	5.0	5.0	4.9	4.8	6.3	6.4	6.3	6.2
Maximum per cent	9.1	8.6	8.6	7.9	9.7	8.4	6.7	7.5	6.6	6.5	6.2	7.0	6.1	6.3	8.2	7.8	7.8	7.6
Minimum per cent	4.6	4.5	4.5	4.0	4.4	4.5	3.3	4.0	3.6	3.7	4.1	3.8	4.0	3.9	5.4	4.9	5.1	5.2
Probable error	0.60	0.56	0.69	0.60	0.75	0.72	0.35	0.68	0.47	0.42	0.39	0.43	0.35	0.27	0.41	0.38	0.41	0.43
1.60 Sp. Gr. Sink:[a]																		
Ash: Average per cent	42.6	36.0	40.2	39.0	37.8	43.8	44.4	51.2	57.2	56.0	34.0	34.0	34.1	43.5	0	47.0	36.3	39.1
Maximum per cent	70.6	71.5	61.5	50.2	57.7	70.4	86.8	85.3	86.4	84.2	57.8	42.5	47.3	59.1	0	67.9	59.9	54.2
Minimum per cent	23.2	24.0	20.0	20.5	22.8	24.0	24.7	25.3	28.8	25.9	23.7	30.0	21.1	23.8	0	38.8	23.3	20.5
Probable error	10.2	12.3	11.2	5.7	8.5	9.7	12.4	10.6	10.9	11.0	8.4	3.7	7.4	8.5	0	8.8	9.2	11.9

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[a] All increments did not contain 1.40 to 1.60 sp. gr. and 1.60 sp. gr. sink fractions. See Table No. 7. The ash probable error of fractions was calculated on basis of the number of increments that contained material.

TABLE 7.—*Variability of Percentage Weight of Specific Gravity Fractions of Individual Sample Increments of 2 to 4-in. Cleaned Coal*

Individual increments taken every 15 minutes during two days loading.

Mine		A					B				C				D			
Test	1	1	2	3	4	5	1	2	3	4	1	2	3	4	1	2	3	4
Weight of each increment, lb.	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25
Number of increments per test	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50
Head Sample, Per Cent	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
1.40 Sp. Gr. Float:																		
Average per cent weight		97.9	97.0	96.7			96.2	95.8			98.2	97.7			98.0	97.6		
Percentage increment with 100 per cent weight		42	24	24			4	4			38	24			28	24		
Minimum with 100 per cent weight		92.3	83.4	89.1			90.1	82.7			93.4	90.7			91.2	90.6		
Probable error[a]		1.80	2.22	2.14			1.65	1.85			1.38	1.57			1.36	1.55		
1.40 × 1.60 Sp. Gr.:																		
Average per cent weight		1.9	2.8	2.7			2.1	2.6			1.7	2.2			2.0	2.3		
Maximum per cent weight		7.5	14.2	10.9			9.7	12.3			5.8	9.3			8.8	9.4		
Percentage increment with no per cent weight		46	28	30			10	10			42	24			30	28		
Probable error[a]		1.94	2.07	1.83			1.14	1.45			1.37	1.46			0.85	1.01		
1.60 Sp. Gr. Float:																		
Average per cent weight	99.0	99.8	99.8	99.4	99.4	99.6	98.3	98.4	98.2	98.8	99.9	99.9	99.9	99.8	100.0	99.9	99.96	99.9
Percentage increment with 100 per cent weight	22	88	82	70	74	80	16	26	14	32	90	92	86	86	100	92	90	88
Minimum with 100 per cent weight	93.4	95.5	95.8	94.8	95.5	96.2	94.8	94.3	90.5	94.1	97.6	97.0	98.9	95.8	100.0	98.7	99.2	98.7
Probable error[a]	0.84	0.27	0.33	0.70	0.78	0.54	1.01	1.23	1.07	0.92	0.17	0.18	0.15	0.31		0.13	0.06	0.13
1.60 Sp. Gr. Sink:																		
Average per cent weight	1.0	0.2	0.2	0.6	0.6	0.4	1.7	1.6	1.8	1.2	0.1	0.1	0.1	0.2	0.0	0.1	0.04	0.1
Maximum per cent weight	6.6	4.5	4.2	5.2	4.5	3.8	5.2	5.7	9.5	5.9	2.4	3.0	1.1	4.2	0.0	1.3	0.8	1.3
Percentage increment with no per cent weight	22	88	82	70	74	80	16	26	14	32	90	92	86	86	100	92	90	88
Probable error[a]	0.83	0.69	0.33	0.70	0.39	0.48	0.77	0.87	1.07	0.61	0.18	0.18	0.15	0.31	0.0	0.05	0.03	0.07

[a] Probable error was calculated on basis of all 50 increments.

TABLE 8.—*Variability of the Weight Per Cent of Specific Gravity Fractions of 2 to 4-in. Cleaned Coal*

1000-lb. gross samples each consisting of 25-lb. increments taken at 15-minute intervals.

Mine	A			B		C		D	
Test	1	2	3	1	2	1	2	1	2
Number of increments per sample	40	40	40	40	40	40	40	40	40
Weight of each increment, lb	25	25	25	25	25	25	25	25	25
Total weight of sample, lb	1000	1000	1000	1000	1000	1000	1000	1000	1000
Specific gravity fractions	Per Cent Weight								
1.28 float	23.1	19.5	30.1	38.3	37.4	52.6	50.9	26.4	25.7
1.28 to 1.31	45.9	65.3	45.0	41.0	41.8	32.3	32.9	43.6	45.1
1.31 to 1.34	22.2	8.1	11.4	9.8	10.4	9.3	9.7	15.2	14.6
1.34 to 1.37	4.5	2.4	4.6	4.9	4.8	1.7	2.9	10.0	9.3
1.37 to 1.40	1.2	1.4	4.7	2.0	1.5	2.3	1.1	3.3	3.4
1.40 to 1.60	2.7	2.9	3.9	2.2	2.2	1.6	2.2	1.5	1.8
1.60 sink	0.4	0.4	0.3	1.7	1.9	0.3	0.3	0.01	0.1

VARIABILITY OF THE PERCENTAGE OF ASH OF SPECIFIC GRAVITY FRACTIONS, 2 TO 4-IN. CLEANED COAL

Mine	A			B		C		D	
Test	1	2	3	1	2	1	2	1	2
Specific gravity fractions	Per cent Ash								
1.28 float	2.9	3.0	2.8	2.8	2.6	3.1	3.0	3.1	3.1
1.28 to 1.31	4.5	4.9	4.7	4.3	4.5	5.3	5.3	5.1	5.2
1.31 to 1.34	7.6	9.5	7.5	7.3	7.7	8.1	8.2	9.3	9.4
1.34 to 1.37	12.9	13.7	10.0	10.9	10.5	11.5	10.7	13.1	12.8
1.37 to 1.40	18.7	22.3	13.1	15.1	14.0	16.5	15.8	15.9	16.3
1.40 to 1.60	28.9	27.8	21.2	21.3	20.5	22.9	22.7	21.1	20.0
1.60 sink	40.8	38.7	46.3	59.0	55.0	54.6	42.1	38.0	44.3

TABLE 9.—*Variability of Fusion Specific Gravity Fractions of Coal from Various Mines with Ash Analysis on One Coal Showing a Large Variation*

Specific Gravity	A	B	C	D	E	F	G	H	I	J	K	Over-all Average
1.30 float	2300	2450	2510	2575	2500	2675	2660	2800	2700	2790	2725	2610
1.40 float	2525	2350	2375	2375	2550	2625	2550	2625	2500	2500	2600	2510
1.50 float	2475	2400	2300	2180	2550	2600	2450	2650	2475	2225	2500	2465
1.60 float	2450	2250	2200	2300	2550	2375	2400	2400	2475	2200	2275	2350
1.70 float	2550	2450	2200	2300	2450	2375	2320	2375	2200	2175	2200	2330
1.70 sink	2425	2450	2225	2550	2375	2500	2340	2575	2175	2450	2400	2400

SAMPLE J

Specific Gravity	Ash Analysis, Per Cent				Fusion Temp., Deg., F
	SiO_2	Fe_2O_3	Al_2O_3	CaO	
1.30 float	51.35	6.97	33.28	3.18	2790
1.40 float	50.75	12.86	29.84	2.63	2500
1.50 float	46.10	12.95	24.00	9.91	2225
1.60 float	44.90	10.67	23.28	9.17	2200
1.70 float	38.75	11.00	19.85	14.88	2175
1.70 sink	54.60	14.54	24.21	2.47	2450

TABLE 10.—*Analyses*

Coal No.	Moisture and Ash Free				Fusion Temperature, Deg. F.	
	Volatile Matter, Per Cent		B.t.u.			
	1 to 2 In.	−200 Mesh	1 to 2 In.	−200 Mesh	1 to 2 In.	−200 Mesh
1	40.5	33.9	14,980	14,700	2,400	2,185
2	38.0	33.0			2,415	2,200
3	37.5	30.6	15,200	15,100	2,500	2,135
4	37.8	29.5	15,250	15,110	2,550	2,175

Fig. 3 shows the maximum observed variations on studies of ash, sulfur, fusion temperature and chemical analyses of ash on float and sink fractions of 0 to 3/8-in. coal. Bright coal, dull coal and fusain have different chemical characteristics. In Table 11 an average of samples taken from 10 mines is shown.

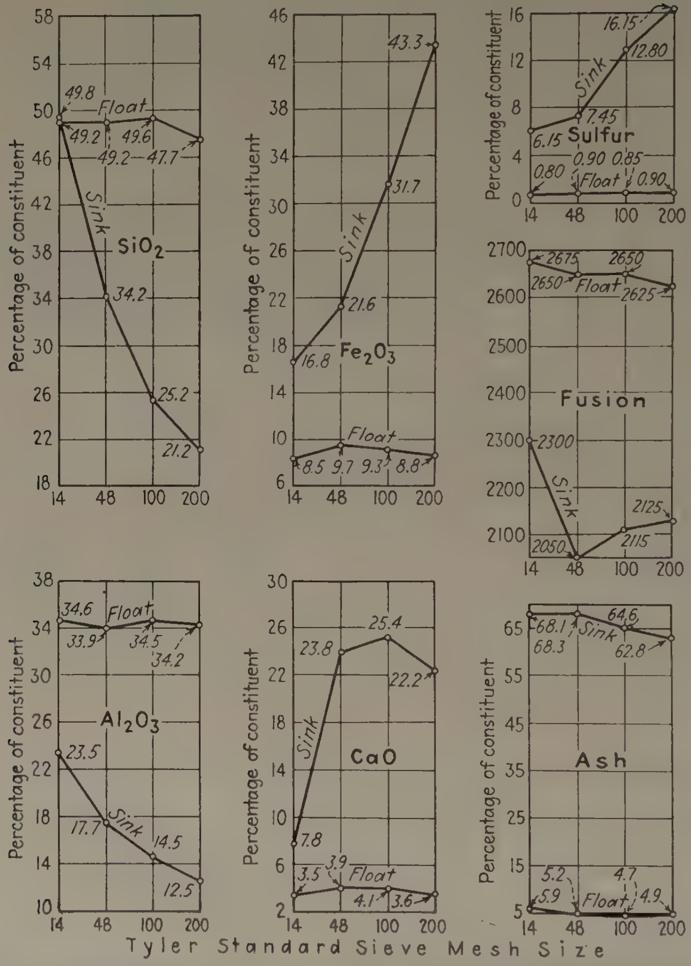

FIG. 3.—MAXIMUM VARIATION IN SIEVE PRODUCTS OF 0 TO ⅜-INCH RAW COAL.

TABLE 11.—*Analyses of Samples from Ten Mines*

	Moisture and Ash Free				Ash Analysis, Per Cent				Fusion Temperature, Deg. F.
	Ash	Sulfur	Volatile	B.t.u.	SiO_2	Al_2O_3	Fe_2O_3	CaO	
Bright coal	2.5	0.80	38.3	15,090	40.9	38.4	11.5	4.0	2620
Dull coal	6.2	0.80	41.9	15,280	54.2	31.6	9.8	2.1	2665
Fusain (soft)	7.3	2.20	10.3	14,600	25.6	26.1	36.9	3.9	2135
Fusain (hard)	29.1	0.90	37.5	11,490	6.0	0.9	6.9	56.0	2500

CAUSES OF VARIATION IN TREATMENT OF GROSS SAMPLE

Crushers

The above discussion has pointed out some of the causes for variation in the taking of the gross sample. Let us now consider what variation we may get in the reduction of the gross sample. At any place where considerable coal sampling is done, a crusher is generally used to reduce the size particles of the sample.

Table 12 shows the results of testing a crusher that produces a product containing about 40 per cent plus 4 mesh. This crusher contains a splitter, which cuts out a portion of the sample after it is crushed and as it falls from the periphery of a bell-shaped crushing segment. It must be borne in mind that the results from this crusher contain in addition the errors of riffling and analysis as well as the errors introduced in splitting the sample in the crusher. The crusher used in this test has a large capacity and is not the small rotary crusher frequently used in crushing coal samples.

TABLE 12.—*Difference in Percentage of Ash between Samples Taken from Sample Spout and from Reject Spout on One Coal Containing* 7 *Per Cent Ash and One Containing* 10 *Per Cent Ash*

Ash, Deviation Between Spouts, Per Cent	Percentage of Samples	
	7.0 Per Cent Ash Coa	10.0 Per Cent Ash Coal
Zero	15.5	10.4
0.3	72.4	
0.5	87.9	50.0
0.8	96.6	
1.0	100.0	70.8
2.0		87.5
4.0		100.0

In some types of crushing units it is possible to obtain constant variations between the sample and the reject spouts, particularly when crushing raw coals. Table 13 shows the results of four series of tests, 10 samples being run for each test. These figures are shown as a horrible example of what can happen; they are not to be considered as representative of normal sample crushing. However, our experience has indicated that the size-weight ratio should be rigidly adhered to in sample reduction and that it is possible to obtain marked variations, especially with high-ash coals, unless the crusher will produce a product at least 95 per cent through 4 mesh.

TABLE 13.—*Four Series of Tests*

Test No.	Sample Spout	Reject Spout		Average Maximum Variation Between the Three Spouts
		Left	Right	
1	8.6	8.9	9.3	1.2
2	8.3	8.5	8.4	0.8
3	10.1	10.9	10.5	1.6
4	9.6	10.4	10.3	1.0

Other series of tests on relatively uniform low-ash coals with crushers that both reduce and split the sample show a satisfactory agreement between the sample and reject spouts. The principal cause of variations between the sample and reject spouts is varying amounts of plus 4-mesh material, which is generally higher in ash than the minus 4-mesh material.

Holmes expresses his conclusions a little differently but with the same meaning when he says:

> There are several possible methods of proceeding, the first being to crush the sample down until it looks as if it would pass ¼ in., which means, in practice, that most of it would pass ½ in., and that the pieces that would not will include a high percentage of hard material of high ash content. One would have to include the "Act of God" and similar clauses in any insurance against dissimilarity between the two halves of a sample crushed down to these limits. And yet, how many people are content with such methods of sampling, either because they do not know, will not appreciate, or are too disinterested to check the differences between the two halves of the sample?

The structure of the coal has a marked effect on the crushing problem. Consider a coal of soft structure and a coal of hard structure, both having the same average amount of impurities in the mine-run. A relatively large amount of soft-structure coal will have broken down and diluted the percentage of impurities in the slack. Thus the problem of sampling and sample reduction is less complicated on this type of coal.

Variations Due to Riffling

In order to study the effect of riffling, two tests were made on the 0 to ⅜-in. coal. A pile of 2000 lb. of coal was coned and quartered into two samples of 1000 lb. each. The first test consisted of riffling 1000 lb. into 256 samples of approximately 4 lb. each. These were then crushed to 10 mesh, riffled three times, and pulverized to 60 mesh. The second sample of 1000 lb. was first reduced to approximately minus 8 mesh and then riffled into 256 samples of 4 lb. each, from which pulps were prepared similar to test No. 1. On both tests the ash content was determined in duplicate on 2-gram pulps.

TABLE 14.—*Study of Riffling*

Each test consisting of 256 samples of 4 lb. each from separate 1000-lb. gross samples of 0 to ⅜-in. coal

DEVIATION FROM AVERAGE ASH OF INDIVIDUAL 4-LB. SAMPLES

	First Test	Second Test
Limits from average:		
± 0.10	48.4	76.0
± 0.20	83.2	97.0
± 0.30	98.8	100.0
± 0.40	100.0	
Probable error	0.094	0.059
Ash: average per cent	6.75	6.79
Maximum per cent	7.12	7.09
Minimum per cent	6.48	6.50
Weight of gross sample, lb	1024	1024
Number of riffled samples	256	256
Weight of each riffled sample, lb	4	4
Size of coal riffled	30% plus 4 mesh	10% plus 8 mesh

Several investigators have made tests on the analytical accuracy of determining ash content in coal. Davis and Fairchance obtained a probable error of 0.043 in analyzing a coal containing 8.9 per cent ash and Fuel Research obtained 0.035 for an ash of 3.7 per cent. Martin and Mandel obtained a probable error of 0.068 on coals between 10.0 and 15.0 per cent ash with a maximum difference on the same sample of 2.15 per cent at different laboratories.

Table 15 shows the differences in ash between duplicate determinations made at two laboratories. Each laboratory made two determinations on each of two series of 256 pulps, riffled from the same gross sample.

TABLE 15.—*Analysis Difference between Percentage of Ash on Duplicate Runs*

Analysis Difference, Per Cent Ash[a]	Percentage of Sample	
	Laboratory No. 1	Laboratory No. 2
0.00	11.7	19.1
0.10	84.0	84.4
0.20	98.0	96.1
0.30	99.6	98.8
0.40	99.6	99.6
0.50	99.6	100.0
0.60	100.0	
Pulps tested	256	256
Probable error of one determination	0.047	0.040
Average ash	6.8	6.8

[a] Ash was determined on 2 grams of pulp.

Some experimental work has indicated that in plant-control laboratories the probable error of an ash analysis can be materially reduced by burning 2 grams of pulp rather than 1 gram.

Effect on Coal Sampling of Recent Changes in Preparation

In 1895 Brunton worked out the effect of the presence or absence of the largest, richest piece of ore for different fineness of crushing and for different specific gravity of minerals.

What Brunton had done on ore, Bailey did on coal by working out the effect of the largest piece of impurity. Bailey also pointed out that the results from coal sampling follow the law of probabilities. It must be remembered that Bailey did most of his work on a coal of 2 mesh and finer that ran 11.5 per cent ash, and that he assumes in his calculations that coal contains 5 per cent ash in the form of slate and other impurities. From these data and further studies on the weight of the largest pieces of impurities in coal that had been screened on bar screens he recommended definite sizes of samples to be taken on different sizes of coal regardless of the characteristics of the coal being sampled.

We might find an excuse for reopening this subject because there has been a marked change in coal preparation since 1909; for instance, at the present time only a relatively small percentage of commercial coal is screened over a bar screen, and, of course, picking tables are commonly used on sizes plus 2 in. and often on plus 1-in. sizes. There was practically no mechanical preparation of commercial coal in 1909, whereas now there is a very substantial amount.

The significance of this change in coal preparation, as it affects coal sampling, is that there has been, on the average, a marked reduction in the percentage and character of the free impurities. When we speak of impurity in this paper we mean material heavier than 1.60 specific gravity.

Hand-picked coal 2 to 4 in. in size will normally contain not over 2 per cent of impurities. The inclusion or exclusion of this free impurity will not affect the ash content by more than 1 per cent. Similarly on hand-picked coal 1 to 2 in. in size the normal amount of impurity will average approximately 5 per cent, which means an increase in ash, due to impurities of 2 per cent. If we take these figures in place of the 5 per cent ash increase as used by Bailey in 1909, we can use his formula to reduce the recommended size of the gross sample.

On mechanically cleaned coal the effect of the impurities on the ash should not be over 0.5 per cent on the sizes plus 1 in. and not over 1 per cent in the sizes minus 1 in. down to a point where the size-weight ratio is not such an important factor.

Many mechanical cleaning plants are doing a better job than these figures would indicate which suggests the possibility of taking smaller

TABLE 16.—*Weight and Ash of the Ten Heaviest Pieces of* 1.60 *Sink from* 2 *to* 4-*in. Cleaned Coal*

Each Test Consisted of 80 Increments of 25 Lb. Each

Mine	A				B				C				D			
Test	1		2		1		2		1		2		1		2	
Piece No.	Wt., Grams	Ash, Per Cent	Wt., Grams	Ash, Per Cent	Wt., Grams	Ash, Per Cent	Wt., Grams	Ash, Per Cent	Wt., Grams	Ash, Per Cent	Wt., Grams	Ash, Per Cent	Wt., Grams	Ash, Per Cent	Wt., Grams	Ash, Per Cent
1	668	32.3	676	31.6	650	85.2	515	34.3	237	39.5	302	39.2	187	23.1	106	57.7
2	444	27.5	382	30.0	611	31.2	431	30.4	190	40.7	167	50.1	177	25.7	101	61.7
3	231	39.9	302	66.2	601	45.2	324	53.6	172	24.8	108	24.2	163	25.9	82	24.1
4	216	62.6	268	42.9	485	24.7	323	42.1	148	58.6	95	45.9	74	22.7	69	33.3
5	204	36.6	230	39.5	399	83.2	302	27.9	138	50.5	91	74.4				
6	198	56.7	223	46.7	360	35.6	278	24.0	100	23.0	82	23.9				
7	192	55.9	200	52.2	350	26.0	272	74.5	96	25.0	81	43.9				
8	182	50.8	185	43.0	339	25.1	259	63.7	90	43.8	45	30.7				
9	171	33.0	171	32.0	332	72.7	252	72.6	88	44.4	41	44.2				
10	166	42.6	167	48.2	322	43.9	248	30.0	85	37.6						
Average of 10 largest pieces	267	40.4	280	41.3	445	47.7	320	43.5	134	39.3	112	43.8	150	24.4	90	46.4
Average of all pieces of 1.60 sink	150	39.0	153	39.0	170	54.6	160	53.3	124	38.6	112	43.8	150	24.4	90	46.4
Total number of pieces of 1.60 sink	37		35		123		68		12		9		4		4	
Percentage of 1.60 sink	0.5		0.6		2.0		1.8		0.2		0.1		0.07		0.05	
Ash added by 1.60 sink	0.2		0.2		1.0		0.9		0.1		0.0		0.0		0.0	
Weight of sample, lb	2000		2000		2000		2000		2000		2000		2000		2000	
Average weight of coal, grams				205												

samples of mechanically cleaned coal than of raw coals, to attain the same degree of accuracy.

Recently we have made a study of the actual weight and ash content of individual pieces of impurity in commercially loaded coal. On sizes 1 to 2 in. and 2 to 4 in., 92 tests were made. The total weight of the samples amounted to about 40 tons. The samples were treated without reduction in size or weight. Separation tests were made on gross samples and also on separate increments at both 1.40 and 1.60 sp. gr. The individual pieces of both the 1.60 sp. gr. sink and the 1.40 to 1.60 sp. gr. material were counted, run for ash, and notes made on the shape of the particles.

Table 16 shows the actual weight and ash content of the sink material. Out of a total of 292 pieces of 1.60 sink material there is only one piece—shown in test No. 1, mine B, weighing 650 grams with 85 per cent ash—that has any real significance as far as the effect that the one heaviest piece has on the gross sample. The chance inclusion or exclusion of this one particular piece on a 100-lb. sample would result in a maximum variation of 1.2 per cent ash. In the same manner the chance inclusion or exclusion of the largest piece, found in test No. 1., mine A, weighing 668 grams with 32.3 per cent ash, would result in a maximum variation of less than 0.5 per cent ash. More of the high-ash pieces are found in the pieces of refuse with lower weight, and it is very seldom that the heaviest piece of refuse is high-ash material.

Some of the conclusions arrived at from this testing are as follows:

1. The ratio of the weight of the heaviest single piece of 1.60 sp. gr. sink material to the average weight of coal (1.40 float) varied at different mines on 2 to 4-in. coal from 0.7 to 3.3 and on 1 to 2-in. coal from 1.8 to 4.1.

2. The ash on the heaviest pieces of 1.60 sink on various tests on 2 to 4-in. coal ranged from 23.1 to 85.2 per cent and averaged 42.8 per cent; on 1 to 2-in. coal it ranged from 32.9 to 81.5 per cent and averaged 49.9 per cent.

3. The ratio of the weight of the heaviest piece of 1.40 to 1.60 sp. gr. material to the average weight of coal (1.40 float) varied at different mines on 2 to 4-in. coal from 2.3 to 3.4; on the 1 to 2-in. coal from 3.1 to 4.9.

4. The ash on the heaviest pieces of 1.40 to 1.60 sink on various tests ranged on 2 to 4-in. coal from 17.5 to 30.0 per cent and averaged 23.2 per cent; on 1 to 2-in. coal it ranged from 14.1 to 28.3 per cent and averaged 18.7 per cent.

5. The statements under conclusions 1 and 3 are in fair agreement with Bailey's 3:1 ratio of the largest pieces of impurities to the average weight of coal, but we do not necessarily agree with his conclusions as to the effect of the heaviest pieces on the sampling accuracy, since the

heaviest piece does not usually contain the highest ash. The number of pieces and the average ash of the pieces must also be considered in establishing sampling standards. It should also be noted that impurities through a bar screen will undoubtedly have a higher piece weight than impurities screened through plates with equivalent round holes. Studies of the 1.60 sink material found in these tests showed that 46.5 per cent consisted of flat pieces, 31.5 per cent of elongated pieces. Only 22.0 per cent of the 1.60 sink material was of a cubical shape.

6. When efficient cleaning results in an extremely low percentage of 1.60 sp. gr. sink material, the effect of the total sink on sampling—not only the largest or heaviest piece of material but all of the 1.60 sink material—is almost negligible.

7. The 1.60 sp. gr. sink material is the important factor in sampling accuracy only when the percentage is high. But when the percentage of sink is low, say less than 1 per cent, the variation in the 1.40 float ash becomes more important.

8. Table 17 shows the actual characteristics of impurities in raw coal separated at high gravities. These tests clearly show that the material of highest specific gravity is definitely not the highest in ash. A drop in fusion is noticed between 1.60 to 2.00 and the 1.60 float. An increase is noticed between 2.00 to 2.60 and the 1.60 to 2.00 and a drop in the 2.60 sink. The drop in the 1.60 to 2.00 is due chiefly to the lime concentration at this gravity. The drop in the 2.60 sink is due chiefly to the high

TABLE 17.—*Variation in Analysis of High-gravity Fractions of 0 to 4-in. Raw Coal*

Four coals separated into 1.60 to 2.00, 2.00 to 2.60 and 2.60 sink specific gravity fractions.

Specific Gravity	Per Cent Wt.	Volatile Material, Per Cent Moisture and Ash Free									Fusion Temperature, Deg. F.
		Ash	Sulfur	Moisture	Air Dry	Free	SiO_2	Al_2O_3	Fe_2O_3	CaO	
Coal No. 1											
1.60 to 2.00	16.2	49.7	7.65	0.4	21.9	43.9	49.0	25.0	20.3	1.5	2370
2.00 to 2.60	74.0	77.0	3.40	1.0	12.5	56.8	60.1	28.8	8.6	0.9	2620
2.60 sink...	9.8	68.7	23.90	0.4	18.2	58.9	32.0	16.6	44.6	2.7	2395
Coal No. 2											
1.60 to 2.00	16.9	48.8	5.20	0.5	21.5	42.4	49.0	25.8	15.4	3.9	2320
2.00 to 2.60	76.5	76.1	5.35	0.8	12.8	55.4	57.5	27.7	10.6	0.7	2580
2.60 sink...	6.6	57.6	31.35	0.2	22.2	52.7	4.6	6.5	85.6	1.1	2495
Coal No. 3											
1.60 to 2.00	20.6	50.4	3.65	0.4	19.7	40.0	48.4	23.7	12.3	5.6	2335
2.00 to 2.60	77.5	79.5	1.90	0.5	12.0	60.0	60.1	28.9	5.5	2.0	2625
2.60 sink...	1.9	57.7	34.75	0.1	22.0	52.1	10.3	7.5	79.2	1.8	2530
Coal No. 4											
1.60 to 2.00	19.7	49.9	2.10	0.4	19.9	40.0	49.7	25.1	8.6	7.2	2240
2.00 to 2.60	78.8	79.3	2.30	0.6	11.8	58.7	57.9	28.0	6.5	2.7	2525
2.60 sink...	1.5	61.5	34.65	0.2	22.3	58.2	11.9	10.3	73.8	1.9	2495

percentage of iron, but there appears to be more iron than can flux with the other constituents.

On these coals the average determined volatile (moisture and ash free) of the impurities is higher than that of the shipped product, due in part to the higher percentage of pyritic sulfur and in part to the slate (clays) that contains residual moisture at 105° C. This indicates that on high-ash raw coal the determined volatile (moisture and ash free) would be somewhat higher than the volatile (moisture and ash free) determined on the cleaned coal made from it.

Standard Coal-sampling Methods

It has been our experience that very little commercial sampling of coal actually follows the standard specifications although such claims are very generally made. The standard gross sample is 1000 lb. with exception of a limit of 500 lb. on fine slack containing a relatively small amount of impurity. This amount is not usually taken, especially if the sample must be prepared manually. Often it is commercially impractical to take as large a sample as is now recommended. We believe that it would be better to reduce the size of the standard sample and recognize the limits of commercial accuracy rather than to report samples as having been taken and prepared under standard methods when actually this has not been done.

One of the standard methods requires that: "The 30-lb. quantity shall be crushed to $\frac{3}{16}$-in. size or to pass a No. 4 sieve, mixed, coned, flattened and quartered. The laboratory samples shall include all of one of the quarters, or all of two opposite quarters as may be required." We seriously question the accuracy of a method that permits the taking of only one of the quarters. Holmes made a very interesting study of the segregation encountered in coning and found that the errors due to segregation might be serious. It might be interesting to quote his conclusions:

> In each case there is certainly the probability of "density segregation"—and what a different aspect the sample cone assumes! After all these years of implicit faith in its homogeneous nature, it turns out to be a whitened sepulchre. Careful flattening of the cone towards one side, or taking alternate quarters rather smaller than they should be, would appear to offer some attractions commercially. It is indeed difficult to understand why we should be advised to cone and re-cone our samples three times before finally quartering them. If a single coning can give rise to such pronounced segregation, would it not be better to leave it at that, or, alternatively, adopt a saner method for dividing our samples?

We made a detailed study on the time necessary to prepare according to the standard a 1000-lb. sample of a relatively hard-structure coal 0 to 4 in. in size. The time spent was approximately 12 hr. As the conditions under which the sample was prepared were nearly ideal, we consider this time to be close to a minimum on this type of coal. See

Table 18. Others have stated that the time for collecting and preparing samples according to this standard method would vary from 13 to 16 man-hours, depending on the friability of the coal. At 60¢ an hour this means a labor cost of $7.20 for preparing one sample. When this cost of the hand preparation of the sample is added to the cost of taking and analyzing the sample, total cost is, in many cases, prohibitive for the commercial sampling of small consignments. There is a definite need for a standard on the mechanical crushing and reduction of samples that considers the size-weight ratio and other factors. Many sample crushers today are not fulfilling the requirements of the size-weight theory of reduction in that 30 to 40 per cent of plus 4-mesh is considered to meet the specification of crushing to 4 mesh. The riffling of a small laboratory sample is not satisfactory on coal that contains 30 to 40 per cent of plus 4-mesh product.

TABLE 18.—*Time Study of Sample Reduction by Manual Method on a 0 to 4-in. Raw Coal*

Operation	Man-hours (Using One Man) Productive, Hr. Min.
Crushing 1000 lb. 0 to 4-in. to minus 1 in.	4:00
Forming long pile	0:45
Dividing minus 1-in	0:15
Crushing 500 lb. minus 1-in. to ¾-in.	2:40
Forming long pile	0:10
Dividing minus ¾-in.	0:05
Crushing 250 lb. minus ¾-in. to minus ½-in.	1:05
Coning and quartering minus ½-in.	0:07
Crushing 125 lb. minus ½-in. to minus ⅜-in.	0:45
Rolling 10 times, coning and quartering minus ⅜-in.	0:10
Crushing 60 lb. minus ⅜-in. to minus ¼-in.	0:50
Rolling 10 times, coning and quartering minus ¼-in.	0:07
Crushing 30 lb. minus ¼-in. to 3⁄16-in.	0:55
Rolling 10 times, coning and quartering minus 3⁄16-in.	0:03
Total reduction time	11:37

Accuracy of Large and Small Samples

A large gross sample does not mean that the resultant analysis truly represents the quality of the coal from which the sample was taken or that it truly represents the sample itself. It must be obvious that in the one large 1000-lb. sample there is just as much opportunity for variation in the progressive reduction of the sample as there is in the reduction of one 10-lb. sample. In practice it is easy to reduce the 10-lb. sample in one stage to minus 10 mesh, but the consideration of cost will not permit this rapid reduction of the total large sample. The large sample is most often reduced in two or more stages and we must consider that after each reduction it is necessary to cut out a new sample. We do not question the fact that a large sample properly taken is more often truly representative of the car or cargo from which it is taken than any one small sample. The question is, does the sample pulp accurately represent the gross sample?

Unfortunately, the size-weight ratio in usual sampling practice as it really is—not as it should be—is often "more honored in the breach than in the observance."

Bailey stated that by taking smaller samples and averaging the separate results less weight of coal may be taken to arrive at the same degree of accuracy. He calculates that the weight of a gross sample must be 1.37 times greater than the total weight of 10 individual samples that have been prepared and analyzed separately and of which the results have been averaged.

Grumell and Dunningham, in British Engineering Standard Association *Bulletin* 403, agree with Bailey that the size-weight ratio must be rigidly adhered to in the reduction of samples. Their recommendation on the size of the sample necessary for a certain accuracy is based on extensive experimental work and takes into account the character of the coal and the purpose, and thus the accuracy, for which the sample was taken. They lay more stress on the number of increments than on the weight of the gross sample. In general our experiments check their conclusions provided it is definitely understood that the average of the individual analyses of individual samples is not necessarily the same as the analysis made from a single gross sample made up from a series of increments or small samples.

In order to observe the variation between a series of gross samples and a series of individual increments taken from the same cars, we have recently run some special tests. In one series 35 to 40 increments were taken and composited into a gross sample for each car; in the other series one single increment was taken from each car. The single increment sample was taken from the first coal going into each car. The gross sample consisted of 35 to 40 increments which were taken at intervals equally timed during the loading of each car. The weight of the increments in each series of tests was 4 lb. when sampling slack coal and 10 lb. when sampling nut-slack. Thus, from each car of slack we had one 4-lb. single increment sample and one sample weighing 160 lb. which consisted of forty 4-lb. single increments. On the nut-slack the samples were taken in a similar manner except that the single increments varied from 35 to 40.

The two samples from each car were then analyzed separately for ash, sulfur and ash softening temperature. The data for these tests are shown in Tables 19, 20 and 21. It was very interesting—and, we confess, rather surprising—to note that the average of the analytical results obtained from the single increments as compared to the average of the samples containing from 35 to 40 increments check exactly on the sulfur determinations, check within 0.1 per cent on ash content, and the maximum variation in ash softening temperature was only 60°. In all 240 individual cars were sampled in this manner, one series of a 100, one of 61, and one of 79.

It is interesting to note the probable error of the difference between the two samples from the same car compared with the probable error of the single and multiple increments of single cars from the average of

TABLE 19.—*Comparison of Variability of Percentage of Ash between Two Series of Gross Samples of Different Weight on the Same Cars*

One gross car sample consisted of approximately 35 increments per car equally timed during the loading of the car and the other gross sample from the same car consisted of one increment.

Size	0 to 1⅛ In.		0 to 2 In.			
Mine	A		B		C	
Number increments per car	40	1	40	1	35	1
Weight of each increment, lb	4	4	10	10	10	10
Number cars tested	100	100	79	79	61	61
Ash: average per cent	8.2	8.2	7.6	7.6	5.9	5.8
Maximum per cent	10.1	9.6	9.1	10.9	6.2	6.5
Minimum per cent	7.0	7.0	6.3	5.7	5.4	5.2
	PERCENTAGE OF SAMPLES					
Limits of per cent						
5.0– 5.9				3	67	75
6.0– 6.9			15	27	33	25
7.0– 7.9	26	37	53	36		
8.0– 8.9	67	55	27	27		
9.0– 9.9	6	8	5	2		
10.0–10.9	1			5		
Deviation from average per cent						
± 0.5	74	65	54	39	100	95
± 1.0	95	95	86	75		100
± 2.0	100	100	100	94		
± 3.0				99		
± 4.0				100		
Higher than average per cent	56	48	48	42	48	48
Lower than average per cent	44	52	52	58	52	52
Probable error	0.36	0.38	0.49	0.71	0.10	0.19
Probable error of difference between samples of same car	0.21		0.67		0.19	

all cars. On the 0 to 1⅛-in. sampling the probable error of the differences of the same car is less than either the single or multiple increments. On the 0 to 2-in. sampling the variations of the single increment from the multiple increments of the same car and from its own average are of the same order.

We realize that there are insufficient data in these tests from which to draw too definite conclusions but the actual results seem to offer some interesting possibilities. We might add that we are now obtaining daily averages on the feed to one cleaning plant by taking six single

TABLE 20.—*Comparison of Variability of Percentage of Sulfur between Two Series of Gross Samples of Different Weight on the Same Cars*

One gross car sample consisted of approximately 35 increments per car equally timed during the loading of the car and the other gross sample from the same car consisted of one increment.

Size	0 to 1⅛ In.		0 to 2 In.			
Mine	3		2		1	
Number increments per car	40	1	40	1	35	1
Weight of each increment, lb	4	4	10	10	10	10
Number cars tested	100	100	79	79	61	61
Sulfur: average per cent	2.10	2.10	1.45	1.45	1.00	1.00
Maximum per cent	2.70	2.55	1.70	2.05	1.15	1.20
Minimum per cent	1.60	1.55	1.15	1.00	0.90	0.90
	PERCENTAGE OF SAMPLES					
Limits of per cent						
0.50–0.95					18	23
1.00–1.45			66	53	82	77
1.50–1.95	28	35	34	46		
2.00–2.45	70	59		1		
2.50–2.95	2	6				
3.00–3.45						
Deviation from average per cent						
± 0.1	43	36	59	56	97	92
± 0.2	70	62	92	77	100	100
± 0.3	85	80	100	91		
± 0.4	96	95		95		
± 0.5	98	99		99		
± 1.0	100	100		100		
Higher than average per cent	47	46	43	51	64	62
Lower than average per cent	53	54	57	49	36	38
Probable error	0.16	0.18	0.09	0.13	0.02	0.05
Probable error of difference between samples of same car	0.11		0.12		0.05	

increments with a total weight of 360 lb. instead of one gross sample of 1000 lb. The averaging of the results from the smaller samples is indicating less variation in the raw coal when we had previously obtained from three times as much sample. In this particular case the handling of six individual samples is actually being done at a lower cost. These six 60-lb. samples are taken at equally timed intervals from 4000 tons of plant feed.

TABLE 21.—*Comparison of Variability of Ash Softening Temperature between Two Series of Gross Samples of Different Weight on the Same Cars*

One gross car sample consisted of approximately 35 increments per car equally timed during the loading of the car and the other gross sample from the same car consisted of one increment.

Size	0 to 1⅛ In.		0 to 2 In.			
Mine	I		J		H	
Number increments per car	40	1	35	1	40	1
Weight each increment, lb	4	4	10	10	10	10
Number cars tested	100	100	61	61	79	79
Ash softening temperature, deg. F. average:	2205	2225	2570	2570	2470	2410
Maximum	2470	2525	2630	2700	2610	2700
Minimum	2020	2070	2520	2450	2350	2210
	PERCENTAGE OF SAMPLES					
Limits						
2000–2090	15	11				
2100–2190	41	39				
2200–2290	21	25				21
2300–2390	14	11			10	24
2400–2490	9	12		11	57	33
2500–2590		2	75	56	32	15
2600–2690			25	31	1	5
2700–2790				2		2
Deviation from average						
± 50	34	31	98	48	66	35
± 100	56	55	100	92	95	57
± 150	82	82		100	100	85
± 200	94	93				95
± 250	99	98				97
± 300	100	100				100
Higher than average			54	51	48	53
Lower than average			46	49	52	47
Probable error	80.5	82.2	19.1	43.4	39.0	79.0
Probable error of difference between samples of same car	44		44		85	

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APPENDIX

This appendix contains tables on the commercial car-sampling of coal by increments as suggested by British Engineering Standards Association publications 403, 404, 420 and 502. Tables 22 to 26 show the results of sampling for ash content and Table 27 the results of sampling for sulfur content. Table 28 shows the results of sampling for ash softening temperature and Tables 29 to 32 the results of sampling for sizing tests. Table 33 shows the results of sampling for moisture content.

Table 34 shows a comparison of theoretical and actual distribution. The weight of the increment depended on the size of coal and in most tests the increments weighed 4 lb. on slack coal and 10 lb. on nut-slack coal. In most tests the number of increments per car was 40 except when the loading was too rapid to obtain this number. In all tests the numbers of increments per car was standardized and they were equally timed during the loading of the car.

It should be emphasized again that the variations are total variations and include those caused by: (1) obtaining the gross sample, (2) the preparation of the gross sample, (3) analytical errors, as well as true variations of the coals, and, further, that they are the result of a particular type of sampling.

TABLE 22.—*Variability of Ash Content of Car Samples,* 4-*in. Metallurgical Resultant*

2410 individual car samples consisting of ten 20-lb. increments equally timed during the loading of each car.

Number increments per car	10
Weight each increment, lb	20
Number of cars	2410
Ash: average per cent	6.0
Maximum per cent	8.0
Minimum per cent	5.0

	NUMBER OF CARS	PERCENTAGE OF SAMPLES
Limits of per cent		
Less than 5.5	48	2.0
5.5	68	2.8
5.6	116	4.8
5.7	203	8.4
5.8	349	14.5
5.9	399	16.6
6.0	434	18.0
6.1	275	11.4
6.2	291	12.1
6.3	73	3.0
6.4	67	2.8
6.5	37	1.5
More than 6.5	50	2.1
	2410	100.0
Deviation from Average per cent		
± 0.5		95.9
± 1.0		99.4
± 1.5		99.8
± 2.0		100.0
Probable error		0.14

TABLE 23.—*Variability of Percentage of Ash in Car Samples*

Individual samples consisted of increments equally timed during the loading of each car.

Mine	A				B				C		1		2		3	4		5	6
Test No.[a]	a	b	c	d	a	b	c	d	a	b	a	b	a	b		a	b		
0 to 2 in:																			
Number increments per car	40				40	1			35	1	40		40			40			
Weight each increment, lb	10				10	10			10	10	10		10			10			
Number of cars	100				100	79			100	61	100		32			34			
Ash: average per cent	7.9				7.5	7.6			6.0	5.8	9.1		10.8			11.0			
Maximum per cent	9.1				9.1	10.9			6.7	6.5	12.3		13.6			13.5			
Minimum per cent	6.5				6.3	5.7			5.4	5.2	7.1		8.9			8.4			
Probable error	0.36				0.47	0.71			0.13	0.19	0.61		0.78			0.83			
1⅛ to 2 in.																			
Number increments per car	40				40				35		40		40			40			
Weight each increment, lb	3				3				2		3		3			3			
Number of cars	100				100				100		100		32			34			
Ash: average per cent	7.5				6.1				5.3		9.1		9.1			9.5			
Maximum per cent	9.4				8.3				5.9		13.1		11.3			12.6			
Minimum per cent	6.1				4.8				4.7		7.2		6.2			7.8			
Probable error	0.59				0.45				0.17		0.66		0.71			0.72			
0 to 1⅛ in:																			
Number increments per car	40	40	1	25	40	40	25	25	35		40	40	40	40	40	40	40	40	40
Weight each increment, lb	7	4	4	2	7	4	2	2	8		7	4	7	4	4	7	4	4	4
Number of cars	100	100	100	100	100	100	113	100	100		100	100	32	100	100	34	70	100	100
Ash: average per cent	8.0	8.2	8.2	8.2	8.0	7.9	7.4	7.9	6.1		9.2	8.9	11.4	11.6	11.9	11.6	11.5	9.3	8.6
Maximum per cent	9.0	10.1	9.6	9.7	10.0	9.1	9.2	9.4	7.1		13.7	12.6	14.8	14.3	18.0	15.1	13.4	11.7	12.0
Minimum per cent	6.7	7.0	7.0	7.3	6.6	5.8	6.2	7.0	5.5		6.9	6.3	9.4	9.4	8.3	7.9	9.3	7.5	6.0
Probable error	0.33	0.36	0.38	0.39	0.57	0.33	0.38	0.32	0.14		0.73	0.66	0.90	0.89	1.36	0.93	0.77	0.60	0.67
⅜ to 1⅛ in.																			
Number increments per car	40	25			40	25	25	25	35		40		40			40			
Weight each increment, lb	4	2			4	2	2	2	3		4		4			4			
Number of cars	100	100			100	100	100	100	100		100		32			34			
Ash: average per cent	7.6	7.4			7.1	7.2	7.5	6.5	5.7		9.2		10.8			11.3			
Maximum per cent	9.2	9.7			9.9	8.6	8.8	7.4	6.2		13.1		14.5			16.0			
Minimum per cent	6.1	6.5			5.5	6.4	6.2	5.6	5.3		6.2		8.2			8.5			
Probable error	0.45	0.44			0.71	0.36	0.35	0.37	0.14		0.72		0.96			0.97			
0 to ⅜ in.																			
Number increments per car	40	25			40	25			35		40		40			40			
Weight each increment, lb	3	2			3	2			5		3		3			3			
Number of cars	100	107			100	100			100		100		32			34			
Ash: average per cent	8.6	7.5			9.2	8.7			6.4		9.1		12.2			12.1			
Maximum per cent	10.0	9.3			11.5	10.7			7.8		14.8		16.8			18.2			
Minimum per cent	7.8	6.5			6.2	7.2			5.7		6.6		9.5			6.9			
Probable error	0.30	0.40			0.61	0.58			0.17		0.92		1.06			1.49			

[a] a, b, c and d indicate different tests. No relationship between the same test number for the different sizes.

TABLE 24.—*Variability of Ash Content of Individual Sample Increments, Minus 1-in. Cleaned Coal*

Individual increments of 2 and 5 lb. taken during (stream sampling) and after (top sampling) loading of four railroad cars. Cars loaded at mine B.

	Stream Sampling		Top Sampling	
	2-lb. Increments	5-lb. Increments	2-lb. Increments	5-lb. Increments
Number increments per car	25	25	25	25
Weight each increment, lb	2	5	2	5
Number of cars	4	4	4	4
Total number increments	100	100	100	100
Ash: average per cent	8.0	8.1	7.3	7.7
Maximum per cent	9.7	9.8	9.1	10.4
Minimum per cent	6.3	5.8	5.7	5.2
	PERCENTAGE OF INCREMENTS			
Limits of per cent				
Less than 6.0	0	2	1	5
6.0– 6.4	1	17	4	8
6.5– 6.9	8	15	1	13
7.0– 7.4	14	21	10	16
7.5– 7.9	19	20	32	17
8.0– 8.4	32	18	28	13
8.5– 8.9	19	5	12	16
9.0– 9.4	5	2	6	9
9.5– 9.9	2	0	6	2
10.0–10.4	0	0	0	1
Over 10.4	0	0	0	0
	100	100	100	100
Deviation from average per cent				
± 0.5	51	38	42	46
± 1.0	84	64	71	75
± 1.5	97	81	87	92
± 2.0	100	98	94	99
± 2.5		100	100	100
Over 2.5				
Probable error	0.50	0.79	0.75	0.68
Percentage of ash in Car No. 1	8.1	8.5	7.7	8.5
No. 2	8.3	8.1	7.2	7.3
No. 3	7.7	7.8	7.2	7.6
No. 4	8.0	7.9	7.1	7.4

TABLE 25.—*Variability of Ash Content of Individual Sample Increments, ⅜ by 1-in. Cleaned Coal*

Individual increments of 2 and 5 lb. taken during (stream sampling) and after (top sampling) the loading of four railroad cars. Cars loaded at mine B.

	Stream Sampling		Top Sampling	
	2-lb. Increment	5-lb. Increment	2-lb. Increment	5-lb. Increment
Number increments per car	25	25	25	25
Weight each increment, lb	2	5	2	5
Number of cars	4	4	4	4
Total number increments	100	100	100	100
Ash: average per cent	7.0	7.2	7.0	7.1
Maximum per cent	11.3	10.6	10.5	10.5
Minimum per cent	5.1	4.9	5.4	5.5
	PERCENTAGE OF INCREMENTS			
Limits of per cent				
Less than 6.0	11	10	13	8
6.0– 6.4	18	21	23	18
6.5– 6.9	22	14	14	25
7.0– 7.4	23	20	23	19
7.5– 7.9	12	13	14	12
8.0– 8.4	10	9	5	6
8.5– 8.9	3	5	4	6
9.0– 9.4	0	4	2	3
9.5– 9.9	0	1	1	2
10.0–10.4	0	2	0	0
Over 10.4	1	1	1	1
	100	100	100	100
Deviation from average per cent				
± 0.5	47	35	40	44
± 1.0	78	62	74	74
± 1.5	94	88	92	80
± 2.0	99	94	96	94
± 2.5	99	97	98	97
Over 2.5	100	100	100	100
Probable error	0.57	0.73	0.65	0.66
Percentage of ash in car No. 1	7.0	7.1	7.5	7.6
No. 2	7.1	7.3	7.0	7.1
No. 3	7.3	7.1	6.5	6.7
No. 4	7.5	7.2	6.9	7.0

TABLE 26.—*Variability of Ash Content of Individual Sample Increments, 0 to ⅜-in. Cleaned Coal*

Individual increments of 2 and 5 lb. taken during (stream sampling) and after (top sampling) the loading of four railroad cars. Cars loaded at mine B.

	Stream Sampling		Top Sampling	
	2-lb. Increment	5-lb. Increment	2-lb. Increment	5-lb. Increment
Number increments per car	25	25	25	25
Weight each increment, lb	2	5	2	5
Number of cars	4	4	4	4
Total number increments	100	100	100	100
Ash: average per cent	8.7	8.3	7.8	7.9
Maximum per cent	13.2	10.3	11.2	11.8
Minimum per cent	6.8	6.6	5.3	5.5
	PERCENTAGE OF INCREMENTS			
Limits of per cent				
Less than 6.0	0	0	6	3
6.0– 6.4	0	0	9	9
6.5– 6.9	2	1	10	12
7.0– 7.4	6	13	15	14
7.5– 7.9	9	23	19	15
8.0– 8.4	22	24	14	20
8.5– 8.9	28	20	10	6
9.0– 9.4	19	13	7	9
9.5– 9.9	9	4	5	5
10.0–10.4	3	2	2	4
Over 10.4	2	0	3	3
	100	100	100	100
Deviation from average per cent				
± 0.5	52	44	33	35
± 1.0	83	80	58	55
± 1.5	93	97	75	76
± 2.0	99	100	89	90
± 2.5	99		97	97
Over 2.5	100		100	100
Probable error	0.48	0.52	0.77	0.83
Percentage of ash in car No. 1	8.4	8.1	7.8	8.1
No. 2	8.4	8.0	6.9	7.1
No. 3	8.7	8.4	8.0	7.8
No. 4	9.2	8.5	8.4	8.5

TABLE 27.—*Variability of Percentage of Sulfur in Car Samples*

Individual car samples consisted of increments equally timed during the loading of each car.

Mine	A		B		C		D	E	F	1		2		3		
Test[a]	a	b	a	b	a	b				a	b	a	b	a	b	c
0 to 2 in.																
Number increments per car		40		40		40				35	1	1	40			40
Weight each increment, lb		10		10		10				10	10	10	10			10
Number of cars		34		100		32				100	61	79	100			100
Sulfur: average per cent		1.85		1.20		2.45				1.05	1.00	1.45	1.45			2.00
Maximum per cent		3.30		1.95		3.10				1.20	1.20	2.05	1.70			2.60
Minimum per cent		1.45		0.85		1.95				0.90	0.90	1.00	1.15			1.55
Probable error		0.23		0.14		0.23				0.04	0.05	0.13	0.09			0.16
1⅛ to 2 in.																
Number increments per car		40		40		40				35			40			40
Weight each increment, lbs		3		3		3				2			3			3
Number of cars		34		100		32				100			100			100
Sulfur: average per cent		1.80		1.10		1.95				0.90			1.30			1.90
Maximum per cent		3.05		1.90		2.55				1.25			1.55			2.75
Minimum per cent		1.40		0.70		1.40				0.75			1.05			1.40
Probable error		0.20		0.14		0.27				0.05			0.08			0.21
0 to 1⅛ in.																
Number of increments per car	40	40	40	40	40	40	40	40	40	35		40	40	40	1	40
Weight of each increment, lb	4	7	4	7	4	7	4	4	4	8		4	7	4	4	7
Number of cars	70	34	100	100	100	32	100	100	100	100		100	100	100	100	100
Sulfur: average per cent	1.90	1.90	1.20	1.20	2.35	2.60	1.10	1.00	2.30	1.05		1.45	1.50	2.10	2.10	2.05
Maximum per cent	3.40	3.55	1.80	1.80	3.20	3.35	1.50	1.55	3.30	1.25		1.85	1.80	2.70	2.55	2.65
Minimum per cent	1.40	1.50	0.75	0.85	1.75	2.00	0.85	0.75	1.70	0.90		1.15	1.20	1.60	1.55	1.60
Probable error	0.26	0.30	0.16	0.13	0.18	0.23	0.09	0.08	0.22	0.05		0.08	0.10	0.16	0.18	0.16
⅜ to 1⅛ in.																
Number increments per car		40		40		40				35			40			40
Weight each increment, lb		4		4		4				3			4			4
Number of cars		34		100		32				100			100			100
Sulfur: average per cent		1.85		1.20		2.45				1.00			1.35			1.95
Maximum per cent		3.60		2.00		3.30				1.25			1.70			2.70
Minimum per cent		1.35		0.80		1.80				0.85			1.10			1.45
Probable error		0.28		0.16		0.27				0.05			0.09			0.20
0 to ⅜ in.																
Number increments per car		40		40		40				35			40			40
Weight each increment, lb		3		3		3				5			3			3
Number of cars		34		100		32				100			100			100
Sulfur: average per cent		2.00		1.20		2.80				1.10			1.70			2.20
Maximum per cent		4.10		1.85		3.65				1.35			2.30			2.70
Minimum per cent		1.45		0.80		2.25				0.85			1.30			2.75
Probable error		0.45		0.15		0.21				0.05			0.12			0.16

[a] a, b and c indicate different tests. No relationship between the same test number for the different sizes.

TABLE 28.—*Variability of Ash Softening Temperature of Car Samples*

Individual car samples consisted of increments equally timed during the loading of each car.

Mine	H			I		J		10		11		12		13	1	15
Test[a]	a	b	a	b	c	a	b	a	b	a	b	a	b			
0 to 2 in.																
Number increments per car	40	1	40			35	1	40		40		40				
Weight each increment, lb.	10	10	10			10	10	10		10		10				
Number of cars	100	79	100			100	61	32		100		34				
Ash softening temp.: average	2460	2410	2400			2550	2570	2360		2580		2480				
Maximum	2610	2700	2505			2640	2700	2500		2670		2660				
Minimum	2350	2210	2220			2400	2450	2190		2380		2200				
Probable error	37	79	35			34	43	53		36		58				
1⅛ to 2 in.																
Number increments per car	40		40			35		40		40		40				
Weight each increment, lb.	3		3			2		3		3		3				
Number of cars	100		100			100		32		100		34				
Ash softening temp.: average	2590		2430			2690		2450		2680		2540				
Maximum	2710		2550			2750+		2600		2750+		2710				
Minimum	2430		2230			2520		2190		2480		2380				
Probable error	51		40			34		46		39		44				
0 to 1⅛ in.																
Number increments per car	40	40	40	40	1	35		40	40	40	40	40	40	40	40	40
Weight each increment, lb.	7	4	7	4	4	8		7	4	7	4	7	4	4	4	4
Number of cars	100	100	100	100	100	100		32	100	100	100	34	70	100	100	100
Ash softening temp.: average	2410	2460	2385	2205	2225	2510		2320	2370	2540	2475	2450	2390	2430	2415	2290
Maximum	2560	2540	2490	2470	2525	2610		2450	2445	2650	2635	2650	2600	2550	2510	2440
Minimum	2300	2370	2215	2020	2070	2330		2150	2235	2330	2335	2140	2130	2320	2290	2080
Probable error	35	22	38	81	82	39		57	32	40	41	68	57	30	30	51
⅜ by 1⅛ in.																
Number increments per car	40		40			35		40		40		40				
Weight each increment, lb.	4		4			3		4		4		4				
Number of cars	100		100			100		32		100		34				
Ash softening temp.: average	2530		2420			2660		2380		2660		2530				
Maximum	2690		2545			2750+		2520		2750+		2710				
Minimum	2400		2235			2470		2190		2460		2190				
Probable error	43		44			35		52		41		69				
0 to ⅜ in.																
Number increments per car	40		40			35		40		40		40				
Weight each increment, lb.	3		3			5		3		3		3				
Number of cars	100		100			100		32		100		34				
Ash softening temp.: average	2250		2330			2420		2230		2410		2330				
Maximcm	2420		2495			2530		2450		2560		2610				
Minimum	2160		2130			2200		2030		2210		2040				
Probable error	31		52			45		76		46		99				

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[a] a, b and c indicate different tests. No relationship between the same test number for the different sizes.

TABLE 29.—*Variability of Percentage Size in Car Samples of 0 to 2-in. Coal*

Individual car samples consisted of approximately forty 10-lb. increments equally timed during loading of each car.

Size	1⅛ to 2 in.						⅜ to 1⅛ in.						0 to ⅜ in.					
Mines	A	B	C	1	2	4	A	B	C	1	2	4	A	B	C	1	2	4
Number of cars	100	100	100	100	32	34	100	100	100	100	32	34	100	100	100	100	32	34
Size: average per cent	31.6	26.9	21.8	29.1	29.1	29.7	42.0	41.7	30.1	37.1	39.3	42.4	26.4	31.4	48.1	33.8	31.6	27.9
Maximum per cent	48.7	48.0	34.4	42.7	35.9	37.6	58.7	53.0	48.8	44.9	45.3	46.1	51.5	44.5	62.9	46.8	38.4	39.7
Minimum per cent	13.2	18.0	14.2	18.0	15.0	18.8	25.7	31.5	7.4	31.7	34.7	39.2	8.1	17.5	34.8	21.5	25.4	20.4
Limits of per cent	PERCENTAGE OF SAMPLES																	
0.0– 9.9	8	4	38	3	6	3			1				1					
10.0–19.9	40	72	58	50	38	53			1				22	4				
20.0–29.9	34	19	4	46	56	44	4		47	31			51	37		31	34	71
30.0–39.9	18	5		1			37	32	50	51	59	15	15	52	7	51	63	29
40.0–49.9							43	64	1	18	41	85	10	7	50	18	3	
50.0–59.9							16	4					1		41			
60.0–69.9															2			
Deviation from average per cent																		
± 5.0	38	75	84	65	84	74	52	79	87	97	94	100	46	55	67	57	69	88
± 10.0	74	91	99	96	94	97	86	96	97	100	100		78	92	91	98	100	97
± 15.0	91	96	100	100	100	100	97	100	98				91	100	100	100		100
± 20.0	100	99					100		99				96					
± 25.0		100							100				99					
± 30.0													100					
Higher than average per cent	45	38	46	52	59	47	46	52	49	54	50	47	41	51	53	44	50	50
Lower than average per cent	55	62	54	48	41	53	54	48	51	46	50	53	59	49	47	56	50	50
Probable error	6.1	3.8	2.5	3.7	3.2	2.9	4.9	3.0	2.5	1.6	1.8	1.3	5.6	4.2	3.7	4.0	2.8	2.2

In order to indicate the skewness of curve and to be of assistance in pointing out the variation from the normal law, some of the data in these tables have been reworked to show separately the percentage of the plus and minus deviations within multiple limits of probable error. Table 34 is a summary of the work on 29 separate tests, sampling approximately 100 cars to each test. On ash and sulfur the greater percentage of deviations is on the minus side. The large deviations (exceeding $4r$) are mostly on the plus side. This seems to us to give a pretty complete picture of the skewness and shows that there is a tail on the plus side for ash and sulfur.

On fusion the curve is opposite to the ash and sulfur curve, the tail being on the minus side. This, of course, can be logically accounted for in that a small amount of low-fusion coal nearly always has a greater effect on the average than the weighted calculations would indicate.

TABLE 30.—*Variability of Percentage Size of Minus ⅜-in. in Car Samples of 0 to 1⅛-in. Cleaned Slack*

Individual car samples consisted of increments equally timed during the loading of each car.

Mine	A	A	B
Year	1930	1933	1930
Number increments per car	25	25	25
Weight each increment, lb	2	2	2
Number of cars	113	100	100
⅜-in. size: average per cent	49	47	45
Maximum per cent	74	70	55
Minimum per cent	31	33	34
	PERCENTAGE OF SAMPLES		
Limits of per cent			
Less than 35	9	5	2
35–39	6	11	3
40–44	17	17	44
45–49	17	38	47
50–54	19	18	4
55–59	15	9	
60–69	12	2	
Over 70	5		
Deviation from average per cent			
± 5.0	36	73	93
± 10.0	68	93	99
± 15.0	86	100	100
± 20.0	95		
± 25.0	100		
± 30.0			
Probable error	6.8	3.9	2.6

TABLE 31.—*Variability of Percentage Size of +4-mesh in Car Samples of 0 to ⅜-in. Cleaned Coal*

Individual car samples consisted of increments equally timed during loading of each car.

Mine	A	B	B
Year	1930	1932	1930
Number increments per car	25	25	25
Weight each increment, lb	2	2	2
Number of cars	107	140	100
+4-mesh size: average per cent	20.0	38.0	40.0
Maximum per cent	28.0	54.0	68.8
Minimum per cent	11.5	20.5	21.8
	PERCENTAGE OF SAMPLES		
Limits of per cent			
Less than 15	5		
15–19	38		
20–24	44	4	3
25–29	13	17	7
30–34		14	21
35–39		14	26
40–44		28	20
45–49		17	10
50–54		6	6
55–59			2
Over 60			5
	100	100	100
Deviation from average per cent			
± 5.0	82	42	56
± 10.0	100	73	83
± 15.0		96	90
± 20.0		100	95
± 25.0			100
Over 25.0			
Probable error	2.8	5.9	6.1
Top size	⅜-in. rd. hole	7/16-in. rd. hole and changes in plant operation	⅜-in. Ton-cap.

TABLE 32.—*Variability of Percentage Size of Minus* 100-*mesh in Car Samples of* ⅜-*in. Cleaned Coal*

Individual car samples consisted of increments equally timed during the loading of each car.

Mine	A	B
Year	1930	1930
Number increments per car	25	25
Weight each increment, lb	2	2
Number of cars	107	100
100-mesh size: average per cent	8.0	6.9
Maximum per cent	12.5	13.8
Minimum per cent	5.0	1.3
	PERCENTAGE OF SAMPLES	
Limit of per cent		
Less than 2.0		2
2.0– 3.9		9
4.0– 5.9	7	27
6.0– 7.9	47	27
8.0– 9.9	35	25
10.0–11.9	8	9
12.0–13.9	3	1
14.0–15.9	0	0
Over 16.0	0	0
Deviation from average per cent		
± 1.0	43	27
± 2.0	80	54
± 3.0	91	79
± 4.0	97	88
± 5.0	100	97
Over 5.0		100
Probable error	1.2	1.7

TABLE 33.—*Variability of Percentage Moisture in Car Samples of 0 to 1⅛-in. Coal*

Each car sample consisted of two separate 8-lb. samples taken during loading of each car, which were run separately for moisture and results averaged to give car-sample moisture.

Mine	2	3
Number samples per car	2	2
Weight each sample, lb	8	8
Number of cars	54	82
Moisture: average per cent	4.75	3.60
Maximum per cent	8.50	9.70
Minimum per cent	2.00	1.20
	PERCENTAGE OF SAMPLES	
Limits of per cent		
1.0–1.9		12
2.0–2.9	10	26
3.0–3.9	19	30
4.0–4.9	28	16
5.0–5.9	20	9
6.0–6.9	15	5
7.0–7.9	4	0
8.0–8.9	4	0
9.0–9.9		2
Deviation from average per cent		
± 1.0	50	54
± 2.0	80	83
± 3.0	96	96
± 4.0	100	98
± 5.0		98
± 6.0		99
± 7.0		100
Higher than average per cent	49	42
Lower than average per cent	51	58
Probable error	1.04	0.99

TABLE 34.—*Distribution of Percentage of Samples*[a] *Within Limits of Plus and Minus Multiples of Probable Error from the Average Ash, Sulfur or Ash Softening Temperature*

Size of Coal: 0 to 2 in., 1⅛ to 2 in. and 0 to 1⅛ in.

Quality	Ash			Sulfur			Ash Softening Temperature			Actual Average	Theoretical
Size	0 to 2 in.	1⅛ to 2 in.	0 to 1⅛ in.	0 to 2 in.	1⅛ to 2 in.	0 to 1⅛ in.	0 to 2 in.	1⅛ to 2 in.	0 to 1⅛ in.		
Number of Tests[b]	8	6	15	8	6	15	8	6	15		
Limit:											
Average:											
To + r	23.5	25.8	23.3	22.1	23.0	19.2	29.1	28.3	27.1		
To +2r	36.6	39.3	38.7	34.7	35.7	33.6	48.4	50.6	43.5		
To +3r	42.9	46.0	45.7	43.1	43.5	41.1	55.1	53.8	50.2		
To +4r	45.8	48.3	47.8	45.2	45.1	43.3	55.9	54.9	51.8		
To +5r	46.1	49.5	48.3	45.7	45.7	44.4	55.9	54.9	51.9		
To +6r	46.2	49.8	48.5	45.8	45.8	44.5	55.9	54.9	51.9		
To +7r			48.6	46.2	46.5	44.5					
To +8r						44.6					
Average:											
To − r	27.5	23.3	27.7	29.1	27.0	30.8	21.0	24.2	23.9		
To −2r	46.9	43.6	45.2	49.0	49.0	50.1	35.9	35.7	39.9		
To −3r	52.4	49.5	50.4	52.4	53.5	54.7	40.8	41.0	45.1		
To −4r	53.6	49.6	51.3	53.8	53.5	55.2	42.4	42.8	47.0		
To −5r	53.8	50.2	51.3	53.8	53.5	55.3	44.0	44.0	48.0		
To −6r	53.8	50.2	51.3	53.8	53.5	55.4	44.1	45.1	48.1		
To −7r			51.4	53.8	53.5	55.4					
To −8r						55.4					
Average:											
To ±r	51.0	49.1	51.0	51.2	50.0	50.0	50.1	52.5	51.0	50.7	50.0
To ±2r	83.5	82.9	83.9	83.7	84.7	83.7	84.3	86.3	83.4	84.1	82.3
To ±3r	95.3	95.5	96.1	95.5	97.0	95.8	95.9	94.8	95.3	95.9	95.7
To ±4r	99.4	97.9	99.1	99.0	98.6	98.5	98.3	97.7	98.8	98.6	99.3
To ±5r	99.9	99.9	99.6	99.5	99.2	99.7	99.9	98.9	99.9	99.6	99.9
To ±6r	100.0	100.0	99.8	99.6	99.3	99.9	100.0	100.0	100.0	99.8	
To ±7r			100.0	100.0	100.0	99.9				100.0	
To ±8r						100.0				100.0	
Percentage of tests in which one or more samples exceeded +4r	38	50	40	38	50	60	0	0	7		
Percentage of tests in which one or more samples exceeded −4r	13	17	7	0	0	13	50	67	47		

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[a] Percentage of samples that were on the limit (multiples of r) were distributed one-half within and one-half outside the limit. Percentage of samples that were on the average were distributed one-half plus and one-half minus.

[b] Each test represents approximately 100 cars, individually sampled and individually analyzed. *Average* amount of free impurities (1.60 sp. gr. sink) of individual coals (not individual cars) vary widely from a minimum of 0.2 per cent to a maximum of 10.0 per cent.

Percentage of samples with plus deviations as related to percentage with minus deviations are: Ash and sulfur, smaller total percentage but larger percentage exceeding 4r; fusion, larger total percentage but smaller percentage exceeding 4r.

DISCUSSION

J. F. Barkley,* Washington, D.C. (written discussion).—Messrs. Morrow and Proctor have put into one paper a wealth of interesting and valuable detailed data on variables in coal sampling. The paper should help many to conceive more properly the nature of the problem of sampling coal and also the correct importance to attach to various coal analyses.

The authors state that it is their belief that it would be better to reduce the size of the standard sample and recognize the limits of commercial accuracy rather than to report samples as having been taken and prepared under standard methods when actually this has not been done. The contention that the 1000-lb. sample is impractical for a considerable number of cases is shared by many. It would seem, however, that any attempt to lower the amount of the gross sample would result in considerable controversy. It appears doubtful that the statements of those whose work cannot be taken as truthful in regard to the 1000-lb. sample would increase in trustworthiness on a smaller sample. At least, the truthfulness might not increase as fast as the inaccuracies of the smaller sample increase. The authors do not state exactly what is meant by the limits of commercial accuracy in this connection.

The whole question of coal sampling and analysis has to be considered from the viewpoint of each problem at hand, whether it be research or the analysis of some delivered car of coal. The practical side of the question always enters; for example, in the purchase of a car of coal the agreement or decision as to what is an analysis of the coal is more important perhaps than whether or not the coal is exactly what the analysis indicates. I have seen coal rejected on 0.1 per cent volatile by analysis. Although the analysis might well have been misrepresentative to that extent, such a procedure was necessary from a practical standpoint, all tolerances having been originally included in the specification limits as set for an acceptable coal.

It would appear that since judgment must be used with any analysis, decreasing the size of the gross sample would not help the judgment.

Perhaps the best solution for this difficulty would be to campaign for more crushers. Such crushers should not be of the type having a splitter but should grind all the coal to about 3⁄16 in., after which the coal could be brought down to laboratory amounts by riffling or quartering. This is the practice used on the Bureau of Mines sampling trucks.

G. B. Gould,† New York, N.Y. (written discussion).—The data presented by Mr. Morrow and Mr. Proctor in this paper bring forcibly to the front the need for revising our viewpoint on coal-sampling methods in commercial practice, the tolerances that should be allowed, and the way in which the results of coal sampling and testing should be interpreted and used.

There is another aspect of this matter which may well be considered, in relation to acceptance of sampling methods that are commercially practicable, and that is the rapid gain in accuracy obtained by averaging the results of a number of samples. Theoretically, the probable error of an average of a number of samples varies inversely as the square root of the number of samples averaged. This raises the question as to whether for most purposes, and particularly for judging the relative quality of two or more coals, it is worth while to try to obtain extreme accuracy of single samples. By accepting the tolerances that are necessary with a commercially practicable method of sampling, a higher degree of accuracy can be obtained by averaging the results of a series of samples than can possibly be obtained, within practical limits, for a single sample.

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We have made a number of investigations of the relation between the probable deviation of individual samples of a given coal and the deviation between the averages of two or more series of samples, and these have all showed a close adherence of actual results to the theoretical improvement in accuracy.

Each of the three comparisons, summarized in Table 35, represents a rather severe test of practice, compared to theory. In each case, a number of different coals were included, and these coals varied among themselves in their characteristic deviation, although in each group the coals were of somewhat similar character and quality. A large proportion of these samples, too, were taken by firemen and operating engineers, guided only by general directions as to the principles of sampling. They represent the results of what might be described as conscientious but not particularly expert sampling.

TABLE 35.—*Deviation of Sample*[a]

	Deviation of Individual Samples	Deviation of Averages (Actual)	Theoretical Deviation of Averages
14 Middle Western bituminous, averages of 20 samples (1195 samples)	2.42	0.58	0.54
No. 3 buckwheat from 11 collieries (2764 samples), averages of 20	3.46	1.01	0.78
20 low-ash eastern bituminous coal) averages of 10 (2000 samples)	0.67	0.26	0.21

[a] Data from laboratory tests made by Fuel Engineering Company of New York.

It is, of course, true that if important engineering calculations of plant performance or the acceptance of a particular shipment depend upon a single sample, the probable error of that sample should be kept at a minimum by sampling methods appropriate to the occasion. It would seem to be desirable, therefore, in considering the establishment of sampling standards better suited to commercial practice, to set up different standards for different clearly defined purposes.

A. C. FIELDNER,* Washington, D.C. (written discussion).—Mr. Morrow has presented some valuable data on the sampling of coal, which are of especial interest to Committee D-5 on Coal and Coke of the American Society for Testing Materials. This Committee is responsible for the present standard method of sampling coal, which is based largely on the fundamental studies of E. G. Bailey and the supplementary experiments made by the Bureau of Mines. This standard method, formulated about 20 years ago, has stood the test of time and is known to give reliable results, but unfortunately it is not used in commercial practice as much as it should be because of the comparatively large initial sample it requires and the labor involved in reducing this sample to the proper size for transmission to the laboratory. The recent work of the British Engineering Standards Association, which indicates smaller samples than specified by the A.S.T.M. method, makes it advisable to take up this problem and determine whether their modifications are suitable for American conditions. It is hoped that others also will make sampling investigations comparing the A.S.T.M. and the British methods, so that ample evidence may be available on which any future revisions may be based.

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T. W. Guy, Charleston, W. Va. (written discussion).—The results of the very extensive study and experimental work carried out by the authors make a valuable contribution to what is known of the variables in coal sampling. I believe that most of those who have made a careful study of coal sampling will agree that this statement of H. F. Dodge[1] is particularly applicable to coal sampling: "Any practical discussion of quality and its measurement must deal with variability. Physical and chemical properties of a material vary from spot to spot. The quality of a finished product varies from unit to unit. The quality of any mass of product in respect to any characteristic, such as tensile strength of steel wire strand, weight of galvanized coating of iron sheets, etc., is fundamentally a frequency distribution—not a single fixed value."

It seems that the authors' suggestion of a general re-examination of coal sampling, using statistical methods, would be of timely importance to the coal industry. The use of these methods in other large industries has increased rapidly during the past few years.

A number of years ago I had a cleaning and coke test costing approximately $5000 rendered practically worthless by discrepancies among the results of samples by two reputable commercial laboratories and a large coke company. Since then extensive investigation and collection of data has convinced me that:

1. The acceptance of the result of a sample as the quality of the lot from which it was taken leads to frequent wrong decisions in preparation, marketing and use of coal, involving in the aggregate waste of money and effort many times the total cost of coal sampling.

2. In many cases the size of the gross sample seems to be less important than the *re-sampling* which occurs in reducing the gross sample for analysis. For example, Table 19 shows a similar range and distribution of results from 100 gross samples of 4 lb. each, as from 100 of 160 lb. each, all from the same well cleaned coal.

Among others the authors call attention to the following points: (1) The importance of variability as well as average quality, (2) that inaccuracies of sampling and analysis cause a more important part of the seeming variations of a coal than is generally realized, (3) that an expression of the limits of accuracy is necessary in coal sampling.

The last paragraph of page 228 states, "In general, regardless of results which may be obtained, there is really very little variation from one day to another in the quality of mine run coal produced from any one mine." The authors explain that this does not apply to single cars or relatively small tonnages, nevertheless some will no doubt be surprised at this statement, in view of the fact that in their experience the results of daily samples vary widely in some cases, and in many cases through a range greater than 1 to 3 per cent.

In many cases the ash results of similar samples from a *single large lot* of a prepared coal show differences almost as great as similar samples from *different* large lots of the same coal. This seems to prove that the sampling fluctuations or errors are large, and the true variability from lot to lot relatively small.

H. F. Hebley,* Chicago, Ill. (written discussion).—Anyone engaged in the preparation of coal for the market has, from time to time, had misgivings regarding the accuracy of the results obtained from the rather dubious methods in vogue. Sampling is too often looked upon as a necessary evil, and therefore it is considered that the expenditure of time and money should be kept to a minimum. It is a common idea that one sample will represent the quality of the coal sampled. Also, the method of reporting analyses to the nearest $\frac{1}{100}$ per cent creates the impression that sampling is a very exact process. As Holmes rather whimsically puts it, "there is still more rejoicing over one 2-lb. sample which gives 2 per cent ash than over the ninety and nine, which having been properly taken, come out at 5 or 7 per cent."

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[1] H. F. Dodge: Acceptance-Rejection Requirements in Specifications. *Trans.* Amer. Soc. Test. Mat. (1934) **34.**

* Advisory Engineer, Coal Preparation and Cleaning, Commercial Testing & Engineering Co.

Coning and quartering has remained the usual procedure; but when the samples have to be prepared by hand, the monotony of the work brings about the adoption of short cuts which inevitably defeat the purpose. The personal element in this method is apparent, and it is very difficult to form a heap of coal in which the various sized particles are evenly distributed.

Because of the errors at present existing, it does seem that the adoption of the statistical method has much to recommend it. As mentioned in the paper, an expression of limits is necessary in the sampling of coal, and some generally accepted sampling standards should be adopted, the cost of which would bear some relation to the product being handled.

Moisture content is also another factor that is rather loosely divided into inherent, surface, or air-dried, with no clear notion of what is meant by each class.

The question of method of taking the sample is of the utmost importance, as errors in this procedure often lead to acrimonious discussion regarding the discrepancies. For the most part, operators do make a sincere effort at "stream sampling" and their samples can be representative. However, as Holmes has pointed out, if segregation has taken place, the samples may be open to question.

And for the consumer, who has to sample coal from the surface of a car, notwithstanding that he takes nine equal increments from a depth 12 in. below the coal surface, there is always the danger of getting unrepresentative results because of segregation and variation of feed during loading.

It would be a happy solution if both the consumer and producer could sample the stream of coal as the consumer's car was being loaded. Because of errors involved in taking the sample, the reduction of the sample should be closely watched so that the initial errors are not augmented.

When it is considered that cleaning plants involving large expenditures are predicated on results that are often fallacious, some more reliable method of obtaining and reporting results would seem to be highly desirable.

The authors are to be congratulated on furnishing the coal-mining industry with such a valuable contribution.

O. O. Malleis,* Cincinnati, Ohio (written discussion).—This paper is a truly outstanding contribution. Workers in this field will appreciate a paper of this type readily available for reference. The authors have hit upon a most important point when they say that the entire sample shall be crushed so that 95 per cent will pass through a 4-mesh sieve before any subdivision is made. This point is much more important than is generally realized.

The data in Tables 19 to 21, inclusive, comparing 4-lb. and 160-lb. samples are really most surprising, as the authors have said. I believe these data indicate that the size of the gross sample need not be as large as is usually thought in order to obtain a representative quantity to be sent to the laboratory, provided, of course, the entire gross sample is crushed so that at least 95 per cent will pass a 4-mesh sieve. I believe that smaller gross sample and fine pulverization of the entire gross sample go together.

Not long ago there was very little power-operated crushing equipment available. At present, however, nearly all large producers and consumers of coal have their own power-operated crushers. The U.S. Bureau of Mines has a very well equipped coal-sampling truck. Others have since built coal-sampling trucks equipped with power-operated crushers. It will, therefore, cause no particular hardship in general to crush the entire gross sample to at least 95 per cent through 4 mesh, whereas to crush the entire sample by hand would have been impractical. There certainly is no question that more representative samples to be sent to the laboratory would be obtained. This finer crushing may also mean that smaller gross samples may be taken.

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I believe the question of finer pulverization of the sample that is sent to the laboratory should also be considered. I rather doubt that the laboratory is keeping pace with the developments in the field with regard to the preparation of samples.

D. R. MITCHELL,* Urbana, Ill. (written discussion).—I wish to commend the authors on their presentation of an old subject in a most thorough and enlightening manner. If coal screening was considered a "mischievous" practice in the early days of coal preparation, coal sampling as commonly practiced today is just as mischievous. Recently, I received analyses of four truckloads of coal of the same size and coming from the same mine and was asked to evaluate these loads. The four analyses showed a dry ash content of 8.7, 10.1, 11.2 and 18.2, with a notation on the latter that the sample was obviously wrong because a piece of slate was observed in the sample. One could reason just as well that the first three were obviously wrong because no large slate pieces were noticed in them.

This paper is an excellent presentation of technical data, comprehensive and thorough, which should have considerable practical utility by those marketing coal and those consuming coal. No doubt most of us have met skeptics as to the value of coal analyses. If the coal man is to market his product intelligently, and that is what must be done to survive with competition as keen as it is, within and from without the industry, the more information of the kind presented by Messrs. Morrow and Proctor he can find, the better equipped he will be to capture his share of the fuel market.

The need for improved mechanical equipment for reduction of samples is apparent. Research in this particular field is needed and could well be presented by manufacturers of such equipment or sponsored by them as an independent research organization.

H. E. NOLD,† Columbus, Ohio (written discussion).—The authors say: "In general, regardless of results which may be obtained, there is really very little variation from one day to another in the quality of the mine-run produced from any one mine," and further, "underground coal varies in the seam both horizontally and vertically. The vertical variation is generally the major factor in any one mine; the horizontal variation is not so often important in one mine though different districts may show wide variation, especially in the sulfur content." With these statements we agree. It might be worth while however, to remember that the horizontal variations in a mine, especially in sulfur content, are at times large enough to be important. After careful face sampling, a few years ago, at least one mine in West Virginia was able to select and sell metallurgical coal, while the general average sulfur content of the mine was too high for this market.

We are indebted to the authors for the information on car sampling. In many instances where coal is sold on analysis the railroad car is the unit; the samples are taken during the unloading of the car and frequently, if not generally, by inexperienced men who have no conception of the importance of sampling, and care less. The result is frequently a poorly selected sample of less than 100 lb. to represent from 50 to 80 tons of coal. Such sampling in Ohio recently, on coal sold to city schools, led the coal producer to challenge the analyses. Results of sampling in the coal bins, obviously impossible of accuracy, showed little correlation with the car samples.

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We would, therefore, like to raise two questions:

1. Should not coal producers and users cooperate to develop a practical stand and procedure to be followed in car sampling, which would be reasonably accurate and would be made part of the contract in selling or buying coal on specifications?

2. The easiest place to get a fair sample is at the mine when the car is being loaded. Should not all samples be taken at the point of origin, perhaps by a certified neutral party jointly paid by seller and buyer?

W. A. Selvig,* Pittsburgh, Pa. (written discussion).—This paper is especially valuable in connection with the coal-sampling problem because it gives so many data pertaining to the various factors that affect sampling. Heretofore very little published information has been available because of the great expense involved in the making of such tests.

Coal sampling may be divided broadly into the three following phases: (1) The collection of the gross sample, (2) the reduction of the gross sample to convenient size for transmittal to the laboratory, and (3) the laboratory preparation of the sample for analysis. The laboratory preparation of the sample is the least difficult of these phases because it can be carefully standardized and supervised in the laboratory. Considerable data are available in regard to the accuracy of the problem.

The collection of representative gross samples is an extremely difficult problem because of variations in size, impurities and conditions of coal handling under which samples must be collected. The accumulation of data regarding this phase of the problem involves a great deal of labor and expense. The standard method of the American Society for Testing Materials specifies the collection of a large gross sample, usually 1000 lb., with a view of applying a factor of safety so the method can be used for all kinds of coal. The reduction of these large gross samples involves considerable expense, therefore it would be very desirable if more economical methods, having the desired degree of accuracy, might be employed.

As mentioned by the authors, the standard hand-method of reducing gross samples involves a large amount of labor because of the relatively large amount of sample to be crushed and the relatively large number of operations specified in the procedure. The personal element enters into these various operations to a large extent, and the tendency of the sampler is to shorten the procedure by modifying a number of the specified stages of crushing, mixing and quartering. In recent years, mechanical methods of crushing and reducing the size of coal samples are being used extensively. These effect a considerable saving of time and if properly done should be more accurate than the standard hand method. Such mechanical methods also minimize the effect of the personal factor. It would appear logical to standardize and recommend the use of such mechanical means of reducing coal samples and to make the present hand method an optional method to be used only for those that have occasion to sample coal at infrequent intervals.

It is feasible to investigate the various phases of any proposed method for reducing gross samples of coal. Our information is very meager as to what happens in the various stages of the present standard hand method. It is questionable whether it would be worth while to undertake the expense and labor involved in investigating the accuracy of this hand method of reduction of samples. It would appear more profitable to investigate and standardize mechanical methods of reduction, as there is no good reason why such methods should not replace the present cumbersome and expensive hand method.

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J. B. Morrow and C. P. Proctor (written discussion).—Messrs. Malleis, Selvig and Guy have agreed that closer attention must be paid to minimizing the error of reducing the gross sample to the portion of the sample that is sent to the laboratory. The authors have done further work along this line since the publication of this paper and are convinced that fine crushing, greater care in the stages of sample splitting, and close attention to the condition of the sample-reduction equipment are vitally necessary in increasing the accuracy of sample reduction. With careful sample preparation, smaller samples will give results of desired accuracy. A method based on smaller samples with greater accuracy of sample reduction will gain wider acceptance in the industry; will permit the taking of more samples at less cost, and will aid both the producer and consumer to obtain a better knowledge of the coal with which they are dealing.

The A.S.T.M. Committee D-5, sub-committee XIII on the Mechanical Sampling of Coal, is undertaking to formulate a specification for smaller gross samples, for finer initial mechanical crushing before any division, and for subsequent reduction to the laboratory sample.

We agree with Professor Nold that in general the most practical place to collect the gross sample is at the mine, except where the consumer crushes his coal before using, as in pulverized-fuel installation and coke plants.

Fusain Content of Coal Dust from an Illinois Dedusting Plant*

By Gilbert Thiessen†

(New York Meeting, February, 1936)

The use of a pneumatic process for removing fine material (usually less than 48-mesh) is increasing in the preparation of stoker-size coal. The fine material, produced as a byproduct, is known as dedusting-plant dust. Since fusain is the most friable ingredient in bituminous coal (it may readily be rubbed to an impalpable powder with the fingers), it tends to accumulate in this byproduct.

While studying the properties of cokes produced by blending Illinois coals with other materials, considerable quantities of fusain were required. Dedusting-plant dust suggested itself as a convenient supply. It became necessary, therefore, to determine how much fusain was contained in a measured sample of dust, how the fusain was distributed with respect to size and specific gravity in the sample, and whether or not it would be feasible or desirable further to concentrate or to recover the fusain from the dust before using it in the blends to be coked. The results obtained indicate the possibilities of production of a high-fusain material.

Previous Work

Lehmann and Stach have developed a process[1,2] for separation of banded bituminous coals into their band ingredients by impact pulverization followed by screening or air separation. They found, as might be expected, that the fusain was concentrated in the very fine material. Prockat and Raute[3] examined samples from dedusting plants operating in the Ruhr. They separated the dust into various screen sizes and determined the composition of each of these sizes in terms of the banded ingredients of the original coal. In the dusts examined by them, the finest material was always high in fusain, and most of the fusain contained in the original samples could be recovered from this fraction.

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* Published with the permission of the Chief, Illinois State Geological Survey.

† Associate Chemist, Illinois State Geological Survey.

[1] K. Lehmann and E. Stach: D.R.P. 526711.

[2] K. Lehmann: Application of Petrography to Coal Preparation. *Proc.* Third Int. Conf. on Bituminous Coal.

[3] F. Prockat and K. Raute: Korngrossen und Gefugezusammensetzung von Windsichterstauben des Ruhrbezirks. *Glückauf* (1933) **69**, 185–194.

They also found that the dust as a whole was much richer both in fusain and in bright coal than was the coal as it existed in the seam.

Experimental Work

From a bulk sample of 45 lb. of dedusting-plant dust, representing normal production from the plant at the Orient No. 2 mine of the Chicago, Wilmington and Franklin Coal Co. at West Frankfort, Ill., a representative sample was removed for chemical analysis upon receipt (Table 1).

TABLE 1.—*Analysis of Original Sample* (No. C-491) *of Dust*

	B.T.U. PER POUND	PER CENT
Moisture		9.6
Volatile matter		27.1
Fixed carbon		53.5
Ash		9.8
Sulfate sulfur		0.03
Pyritic sulfur		0.72
Organic sulfur		0.44
Total sulfur		1.19
Heating value	11,746	
Unit-coal heating value	14,755	

Preliminary investigations showed that it would be extremely tedious to estimate the fusain content of the sample as received, because of the wide range of particle sizes. Therefore portions of the sample were separated into fractions (1) according to size, and (2) according to specific gravity. These were examined microscopically and the amount of fusain in each fraction was estimated. Knowing the percentage of fusain in each fraction and the percentage of each fraction in the total sample, it was a matter of calculation to find the amount of fusain in the bulk sample.

Method of Size Separation

A representative portion of the sample was air-dried in the standard A.S.T.M. air-drying oven, and 200 grams of this dried material were separated according to size, using U.S. Standard Series sieves of 20, 40, 60, 80, 100 and 200 mesh. A Ro-tap screening machine was used, the length of screening time being 15 minutes. The fractions so obtained were subjected to chemical analysis and to microscopic examination. The results of the screen and chemical analyses are presented in Table 2, and the screen analysis is shown graphically in Fig. 1.

The sized fractions show lower calorific values than they should on the basis of the calorific value of the original sample. This discrepancy can be attributed to exposure during sieving and to the lapse of time between the analyses. Coal samples in fine sizes are very susceptible to degradation. The analyses of the fractions, however, are comparable. The high fusain content of the minus 200-mesh fraction, to be shown later, is reflected in the low volatile-matter content of this fraction. One would expect the unit-coal calorific value of this fraction to be the highest of the series. The fact that it is not may or may not be due to the greater degradation of the very fine material on exposure. Considerable difficulty was encountered in determining the heats of combustion of several of these fractions, owing to their tendency to blow out of the capsule in the calorimeter bomb when ignited. However it is believed

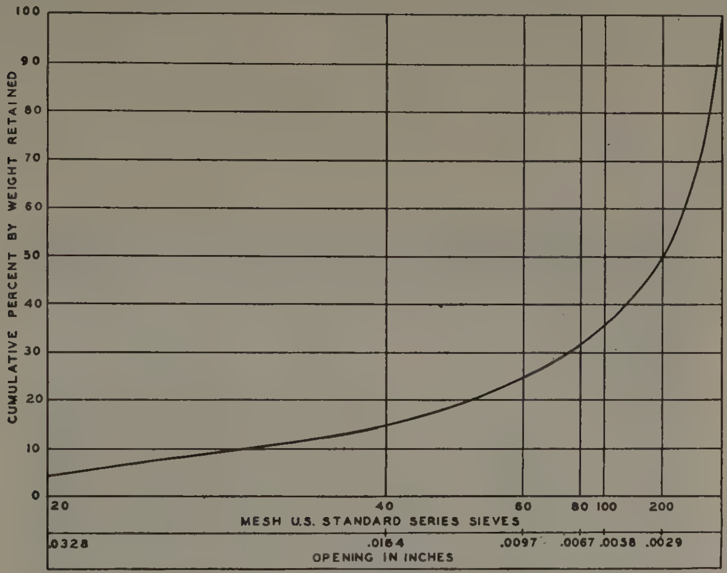

FIG. 1.—CUMULATIVE DIRECT DIAGRAM OF SCREEN ANALYSIS ON SAMPLE C-491 OF DUST FROM DEDUSTING PLANT.

TABLE 2.—*Analyses of Sized Fractions of Dust from Orient No 2*

Sample	Percentage of Size in Sample	Lab. No.	Ash, Per Cent	Volatile Matter, Per Cent	Fixed Carbon, Per Cent	Sulfur, Per Cent				Heating Value, B.t.u. per Lb.	
						Sulfate	Pyritic	Organic	Total	Dry	Unit Coal
Original......	100.0	C491	10.8	30.0	59.2	0.03	0.80	0.49	1.32	12,993	14,755
+20 mesh...	4.4	C717	6.7	36.2	57.1	0.02	0.48	0.55	1.05	13,438	14,519
− 20, + 40.	10.8	C718	9.0	35.7	55.3	0.01	0.62	0.56	1.19	13,055	14,499
− 40, + 60.	9.4	C719	11.7	34.7	53.6	0.01	0.74	0.56	1.31	12,634	14,507
− 60, + 80.	7.2	C720	11.9	34.2	53.9	0.01	0.85	0.48	1.34	12,708	14,629
− 80, +100.	3.8	C721	11.8	34.0	54.2	0.01	0.94	0.40	1.35	12,613	14,502
−100, +200.	14.1	C722	13.5	33.1	53.4	0.02	1.28	0.29	1.59	12,615	14,826
−200 mesh..	50.3[a]	C723	11.3	23.5	65.2	0.04	0.85	0.43	1.32	12,848	14,680
							Weighted average unit coal.				14,647

[a] Includes approximately 0.15 per cent sieving loss.

that the error in this determination has been reduced to a negligible quantity by careful experimental technique.

In preparing series of sized fractions for detailed study, separations were made only on the 50, 80, 100 and 200-mesh screens, since preliminary microscopic examination had shown that only these sizes contained important quantities of fusain.

Estimation of Fusain Microscopically

Small portions of the sized fractions were placed on microscope slides, and the proportion of the fusain particles was estimated by counting. These determinations were made by two investigators, each result being an average of a large number of counts. From these data the weight per cent of fusain could be calculated. In this calculation it was found unnecessary to take into consideration differences in the specific gravities of the fusain particles and the remaining coal particles. However, it is

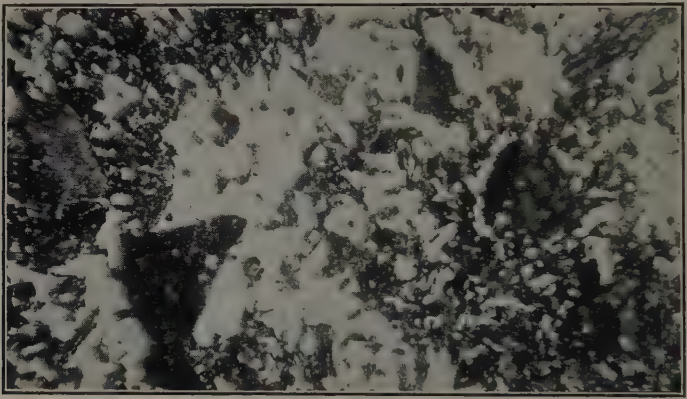

Fig. 2.—Unsized sample of coal dust. × 50.

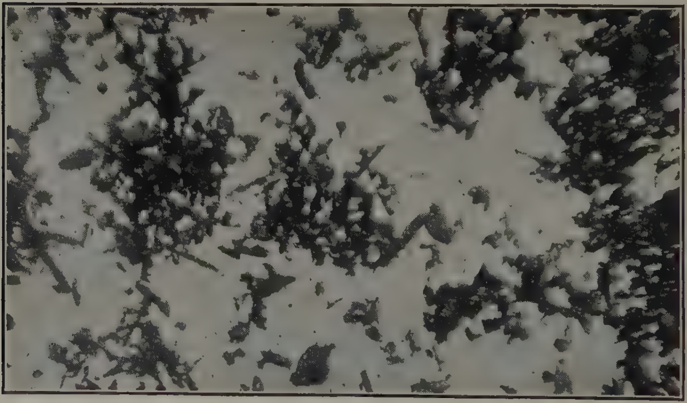

Fig. 3.—Minus 200-mesh fraction. × 50.

known that larger fusain specimens may differ widely in their specific gravities, depending upon their impregnation with mineral matter. These differences disappear to a considerable extent when the material is finely ground and loses much of its entrapped mineral matter. Table 3 shows that 86 per cent of the fusain is contained in the minus 200-mesh material, which makes up approximately one-half of the sample, and that this minus 200-mesh material is approximately three-fourths fusain. The appearance of two of the fractions used in this study are shown by photomicrographs, Figs. 2 and 3.

TABLE 3.—*Results of Estimation of Fusain in Sized Fractions Microscopically*

Fraction	Percentage of Total Sample in Each Fraction	Fusain Content of Fraction, Per Cent	Percentage of Total Sample Occurring as Fusain in Each Fraction	Percentage of Total Fusain in Each Fraction
+50 mesh	22.4	9.7	2.2	5.3
50 × 80 mesh	10.6	8.0	0.8	1.9
80 × 100 mesh	3.8	11.1	0.4	1.0
100 × 200 mesh	14.6	16.3	2.4	5.8
−200 mesh	49.1	72.5	35.6	86.0
Total sample			41.4	100.0

Effect of Separation by Specific Gravity on Fusain Concentration

Since the various banded ingredients of coal have different specific gravities, it should be possible to effect a separation of fusain from the remainder of the dust by a float-and-sink procedure in liquids of different densities, or by some other procedure based on a difference in specific gravity. The experimental results, however, were not encouraging because of the difficulties encountered in separating such finely divided material. The separation of fusain was not as satisfactory as that obtained by sieving. Even a fair estimate of the fusain content of the various gravity fractions could not be made without further sizing.

In this test, a sample of approximately 400 grams, representative of the dust, was separated by float-and-sink procedure with organic liquids of specific gravities 1.3, 1.4, 1.5 and 1.7. Almost 90 per cent of the material was contained in the fractions of 1.3–1.4 and 1.4–1.5 sp. gr. These fractions had to be sieved to make possible an estimate of their fusain content. The results of this work are presented in Tables 4 and 5.

The estimate arrived at by this procedure, while not regarded as quite as reliable as that reached by using size separation alone, checks reasonably well the first value obtained.

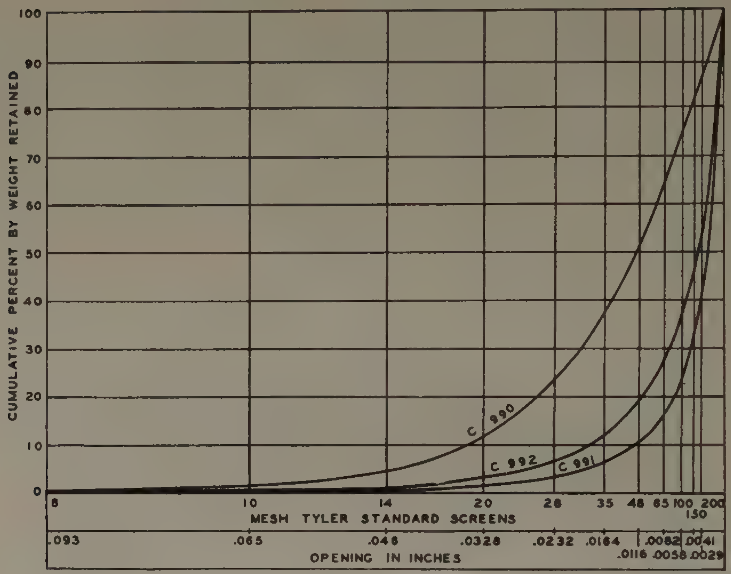

Fig. 4.—Cumulative direct diagram of screen analysis of samples C-990, C-991 and C-992 of dust from dedusting plant.

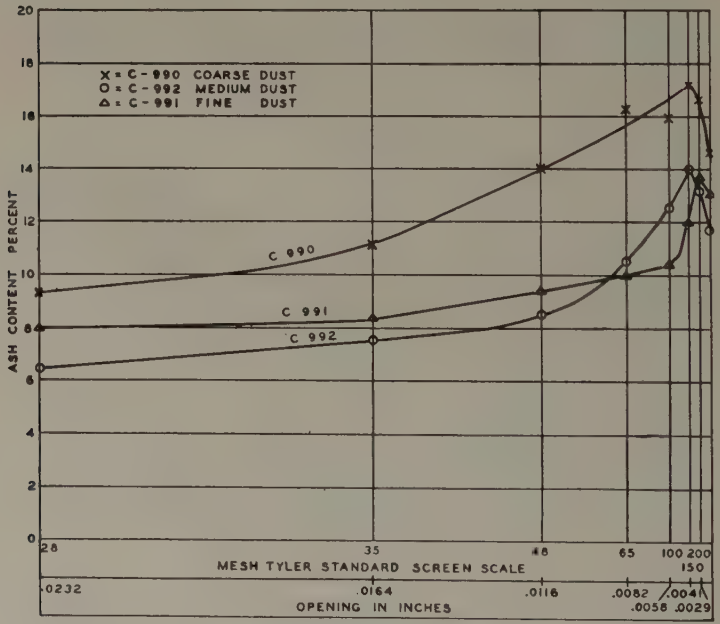

Fig. 5.—Ash content of sized fractions of dedusting-plant dusts, C-990, C-991 and C-992.

TABLE 4.—*Composition of Fractions Separated by Gravity*

Fraction		Percentage of Total Sample in Each Fraction	Fusain Content of Fraction, Per Cent	Percentage of Total Sample Occurring as Fusain in Each Fraction
F 1.3		4.1	nil	nil
F 1.4	S 1.3	56.5	42	23.7
F 1.5	S 1.4	30.7	70	21.5
F 1.7	S 1.5	3.2	50	1.6
	S 1.7	5.5	3	1.7
Total sample				48.5

TABLE 5.—*Size Separation and Fusain Estimation for Fractions of* 1.3–1.4 *and* 1.4–1.5 *Specific Gravity*

Size	Fraction of 1.3–1.4 Sp. Gr.			Fraction of 1.4–1.5 Sp. Gr.		
	Percentage of Sample in Fraction	Fusain Content of Fraction, Per Cent	Percentage of Sample Occurring as Fusain in Each Fraction	Percentage of Sample in Fraction	Fusain Content of Fraction, Per Cent	Percentage of Sample Occurring as Fusain in Each Fraction
+80 mesh	31.5	1	0.3	6.8	30	2.0
80 × 100 mesh	5.4	1	0.1	1.0	30	0.3
100 × 200 mesh	14.6	20	2.9	5.2	50	2.6
−200 mesh	48.5	80	38.8	87.0	75	65.3
Total	100.0		42.1	100.0		70.2

NATURAL RECOVERY OF VERY FINE DUST

Since the work described indicated that the fusain was contained mainly in the minus 200-mesh material, a further natural concentration of fusain could be obtained if the very fine material were collected separately. In the plant from which the dust investigated was obtained, dust is separated out of the air stream at three places: (1) coarse material from the fan casing, (2) finer material from a cyclone separator, and (3) the finest material from a second cyclone separator. Samples of these three materials were obtained and subjected to screen and chemical analysis. The results of these tests are presented in Tables 6 and 7. The size distribution of the samples is shown graphically in Fig. 4. The results of ash determination on the sized fractions are also shown in Table 7 and in Fig. 5. Approximately one-half by weight of the two finer samples consists of minus 200-mesh material. Examination of this fine material indicated that it was composed mainly of fusain. The ash content of the size fraction in a given series rises regularly with decrease in size to a maximum at about +150 mesh and then

TABLE 6.—*Analyses of Coal Dusts from Various Places in Dust-recovery System (Dry Basis)*

Sample No.	C-990	C-992	C-991	C-491
Sample received	11-13-34	11-13-34	11-13-34	11-10-33
Origin	Fan casing	First cyclone	Second cyclone	Total
Ash, per cent	13.2	11.4	12.6	10.8
Volatile matter, per cent	33.2	28.9	27.9	30.0
Fixed carbon, per cent	53.6	59.7	59.5	59.2
Total sulfur, per cent	1.73	1.46	1.68	1.32
Pyritic sulfur, per cent	1.17	0.95	1.12	0.80
Organic sulfur, per cent	0.56	0.50	0.52	0.49
Calorific value, B.t.u. per lb	12,463	12,823	12,705	12,993
Unit coal, B.t.u. per lb., dry	14,597	14,675	14,767	14,755
Unit-coal fixed carbon	62.9	68.5	69.3	67.3
Ash fusion temperature, deg. F.	2155	2121	2090	

TABLE 7.—*Screen Analyses of Coal Dusts from Various Places in Dust-recovery System and Ash Content of Fractions*

TYLER STANDARD SCREEN SERIES USED

Sample, Mesh	C-990 Fan Casing			C-992 First Cyclone			C-991 Second Cyclone		
	Percentage of Total	Percentage of Total Cumulative	Ash Content of Fraction, Per Cent	Percentage of Total	Percentage of Total Cumulative	Ash Content of Fraction, Per Cent	Percentage of Total	Percentage of Total Cumulative	Ash Content of Fraction, Per Cent
On 3	0	0		0	0		0	0	
4	0.07	0.07		0	0		0	0	
6	0.12	0.19		0	0		0	0	
8	0.26	0.45		0.03	0.03		0.01	0.01	
10	1.04	1.49		0.23	0.26		0.10	0.11	
14	3.06	4.55		0.79	1.05		0.40	0.51	
20	7.73	11.78	8.2	1.99	3.04		1.03	1.54	
28	11.58	23.36	9.3	3.42	6.46	6.4	1.96	3.50	7.9
35	14.18	37.54	11.1	5.12	11.58	7.5	2.84	6.34	8.3
48	14.02	51.56	14.0	7.04	18.62	8.5	4.08	10.42	9.4
65	12.65	64.21	16.3	8.25	26.87	10.5	5.18	15.60	10.0
100	11.33	75.54	15.9	9.92	36.79	12.5	8.05	23.65	10.4
150	6.44	81.98	17.2	7.52	44.31	14.0	7.79	31.44	12.0
200	5.17	87.15	16.6	8.97	53.28	13.2	10.70	42.14	13.8
Through 200	12.85	100.00	14.6	46.72	100.00	11.7	57.86	100.00	13.1

decreases, though not to as low a value as that for the fractions of about 48 mesh or larger. To investigate further the change in specific gravity of the dust with size, the specific gravities of the original samples C-990 and C-991 and of certain sized fractions of these samples were determined. The specific gravities were determined by the pycnometer method at 25° C., using benzene as the liquid. The results, which further support our neglect of specific gravity in the estimation of fusain content by microscopic examination, are presented in Table 8.

TABLE 8.—*Specific Gravities of Coal-dust Fractions*

DATA AT 25° C. WATER = 1.00

DESCRIPTION OF SAMPLE	SPECIFIC GRAVITY
Sample C-990	1.41
48 × 65 mesh fraction from C-990	1.40
−200 mesh fraction from C-990	1.51
Sample C-991	1.47
48 × 65 mesh fraction from C-991	1.37
100 × 150 mesh fraction from C-991	1.41
−200 mesh fraction from C-991	1.49

PRACTICAL APPLICATION OF FUSAIN-RICH DUSTS

Mott and Wheeler have shown[4] that the addition of fusain in moderate amounts (usually about 3 per cent) to by-productoven charges of coking coals having strong coking properties permits the production of blockier coke in larger sizes. This effect, it is thought, results from the prevention of shrinkage-strain accumulation in the coke by uniformly releasing them throughout the mass in the form of very small fissures at each fusain particle. Coke dusts and other inert materials, it is claimed, are not as effective, because they are not as completely wet and absorbed into the plastic coking mass as is the fusain. For coals with poor caking strength, the addition of fusain results in a weaker and more sooty coke; its removal from the charge may be beneficial.

Recent work[5] indicates that the naturally formed fine sizes of coal tend to have a higher phosphorus content than does the entire coal. For metallurgical purposes it is often important that the phosphorus content of the coke be very low. The addition of fine coal of high phosphorus content to a coke-oven charge would be undesirable if a low-phosphorus coke is wanted. The removal of the fine, high-phosphorus dust, on the other hand, would be of definite value in an effort to produce coke with as low a content of phosphorus as possible. The phosphorus contents of the total dust sample, of the minus 200-mesh material, and of a corresponding face sample were determined. The results of these determinations are presented in Table 9, which shows that the phosphorus content of the dedusting-plant dust is three times as high as that for the face sample.

This work, showing that the fine sizes of Illinois coals tend to contain large quantities of friable and noncaking fusain, indicates that coke of better structure and lower phosphorus content may be produced

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[4] R. A. Mott and R. V. Wheeler: Brit. Pat. 351546, accepted June 29, 1931.

R. A. Mott and R. V. Wheeler: Coke for Blast Furnaces. Reinhold Pub. Corp. N. Y. 1930.

[5] E. Hoffman and H. Lehmkühler: Untersuchungen über die Phosphorgehalte in Kohlen und Koks und Möglichkeit ihrer Verminderung durch Zweckensprechende Massnahmen. *Brennstoff-Chem.* (1934) **15**: 381–86.

R. Kattwinkel: Die anorganischen Bestandteile der Faserkohle. *Glückauf* (1934) **70**, 1006–9.

Table 9.—*Phosphorus Content of Face Sample, Dedusting-plant Dust and Minus 200-mesh Fraction, Dry Basis*

Sample Description	Laboratory Number	Phosphorus in Ash, Per Cent	Ash in Coal, Per Cent	Phosphorus in Coal Sample, Per Cent
Face sample................	C-1204(C-11)	0.04	10.7	0.004
Total dust.................	C-1205(C-491)	0.11	10.8	0.012
-200-mesh dust...........	C-1206(C-723)	0.07	11.3	0.008

from the fine sizes of Illinois coals if they are dedusted before being coked.

So little has been done in the field of separating the banded ingredients of coal on a large scale that our knowledge of their commercial adaptability is very small. It is to be hoped that the convenient source of fusain concentrate described in this paper may induce other investigators to study its practical usefulness.

Summary

1. Because of its friable nature, fusain is concentrated in the fine coal dust, principally in that of a size less than 200-mesh.
2. The very fine dust, minus 200-mesh, produced during the preparation of coal, is composed mainly of fusain.
3. When dust is recovered during the preparation of stoker fuel in which a dedusting process is used, the dust may serve as a convenient source of fusain on a large scale.
4. A concentrate rich in fusain may be obtained: (1) by recovering the dust in the last of several consecutive stages, or (2) sizing recovered dust by screening or air separation.
5. The coking properties of the fine sizes of Illinois coals may be improved by dedusting, in which process the friable and non-caking fusain is largely removed. The phosphorus content of the coal and consequently of the coke, is reduced at the same time.

Acknowledgment

This work was carried on in the laboratories of the Geochemical Section of the Illinois State Geological Survey, under the direction of Dr. Frank H. Reed, Chief Chemist, whose cooperation during the course of the work is gratefully acknowledged. The author also acknowledges his indebtedness to P. E. Grotts for assistance in carrying on the laboratory work; to John W. Teter and Bazil Bilder, who assisted in the laboratory while employed by the C.W.A.; and to Dr. O. W. Rees, under whose supervision the analytical work was done by J. W. Robinson and C. S. Westerberg. The author wishes to thank Dr. G. H. Cady and Mr. L. C. McCabe for their comments on the manuscript, and to specially thank Mr. T. L. Garwood for his courtesy in supplying the samples.

DISCUSSION

(*Cadwallader Evans, Jr., presiding*)

A. C. Fieldner, Washington, D.C.—In the literature there are many conflicting statements about the effect of fusain. One writer says that fusain is deleterious to the production of good coke; another says that the addition of fusain to a coal improves the physical properties of the coke. Both statements are correct. The action of fusain varies in different coals. The addition of fusain to a highly fusible coal improves the resulting coke, whereas a weakly coking coal such as the high-oxygen Illinois coals will not permit the addition of any fusain without a deleterious effect on the coke. These conclusions apply to the addition of pulverized fusain. Fusain present in hard, flat layers, not thoroughly crushed, introduces planes of weakness in the coke, causing it to break apart and degrade in size. We must look upon the action of fusain in coke formation as an individual problem peculiar to each particular coal and fusain.

The determination of the fusain content of coals is likely to become more important in the future when coal is used as a source of motor fuel produced by the Bergius process. It is probable that fusain cannot be hydrogenated and liquefied, and that it should be removed from coal before an attempt is made to hydrogenate the coal.

I should like to ask Dr. Thiessen whether he tried the electrostatic method for determination of fusain and whether the results obtained by that method were satisfactory.

G. Thiessen.—We did try the electrostatic method. We built a high-voltage, direct-current generator, consisting of a high-voltage transformer, rectifier, voltage divider and a shaking device like that pictured by Davis and Younkins[6]. After a careful study we doubted whether we were getting what we hoped to get. We found that by varying the voltage applied to the plates, we could get off varying quantities of material, and that this material varied in composition. The separating action is due to the charging of the fusain particles because of their conductivity, and noncharging of the coal particles. The charged particles are attracted from the plate on which they are resting to the upper plate, and are then swept through, but the smaller particles of nonfusain material will react somewhat as the larger fusain particles do. We could remove all of the material if we applied high enough voltage, or none if we applied a low voltage.

W. Fuchs,* State College, Pa.—What is fusain? How can we establish the presence of it, or rather, the amount present in a certain coal? We have compared various methods and found no method available that is satisfactory. The petrographic method is somewhat tedious and takes much time, and perhaps does not give excellent results, and the electrostatic method is highly ingenious but, since it involves high voltage and a good deal of expense, is not advisable for general use.

In the literature there is a method mentioned which consists of treating a coal sample with an oxidizing agent—Schultze's solution. By this method practically the bulk of the coal is destroyed and only fusain remains behind. It is one of the features of fusain that it is much less than all the other banded constituents of the coal.

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[6] J. D. Davis and J. A. Younkins: Electrostatic Method for Determining Fusain in Bituminous Coal. *Ind. & Eng. Chem.*, Anal. Ed. (1929) **3**, 165–167, especially Fig. 3. This paper was reprinted as *Cooperative Bull.* 55, Min. and Met. Investigations, Carnegie Inst. of Tech.

* Research Associate Professor, School of Mineral Industries, The Pennsylvania State College.

Therefore, we started a study of that method. We have a Chinese graduate student, Mr. Hsiao, working on this problem. He has studied the effects of all the variables in that method and found certain possibilities to cut down the time and get a rather inexpensive and quick way in which to establish with a first approximation the amount of fusain present.

I do not want to go into the possible errors involved in other methods, but I should like to point out a few that may be involved in the application of chemical methods. The method is based, as has been said, on the fact that fusain does not react as quickly as other constituents of the coal, but there are certain mineral constituents of coal, and some organized plant particles, that might behave in the same way as fusain. When we have a very quick method that is agreeable and easy to handle, certainly there will accumulate more knowledge about the subject than is available at the time being.

Need for Coal Research

By H. H. Lowry,* Member A.I.M.E.

(New York Meeting, February, 1936)

Science attracts the attention and interest of an individual or an industry in general only in proportion to the apparent direct application to its immediate welfare or benefit. Engineering accomplishments, as applied science, therefore receive more recognition than the scientific discoveries on which they are based. Even self-evident truth needs endless repetition before recognition: engineering as applied science cannot advance beyond the science on which it is founded. Scientific research, and discovery, is sometimes forced by the limitations recognized by those active in its development for the use of society and industry and sometimes results from an apparently superficially illogical curiosity about natural phenomena. Research has received inadequate support except in industries born in the laboratory, the chemical and electrical industries, from which many lessons might well be drawn by their elder associates. The rapid growth of these industries has been largely due to close cooperation of scientists and engineers. Most rapid advance is made when the research worker and the engineer keep in close touch with each other's ideas and problems.

Many of the developments of modern industry are based on discoveries buried for years in scientific journals because the "time was not ripe" at their discovery. The situation with regard to utilization of coal is quite the opposite. The fundamental facts in regard to the scientific aspects of the problem of developing the energy from coal are scarcely known. The engineer, without much scientific cooperation, has accomplished much in increasing the number of kilowatt-hours per pound of coal from 0.31 in 1919 to 0.69 in 1934, in increasing the 1000 gross ton-miles of freight hauled from 12.2 to 16.4 per ton of coal in the same period, in fuel economy in production of coke, and in developing the production of motor fuel in Germany and England by hydrogenation of coal.

This lack of scientific research on the utilization of our most important natural resource, coal, has been due undoubtedly in large part to the reluctance of investigators to study the interrelationships involved in the complicated processes in which coal is used. These complexities are made

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more difficult to unravel by the undetermined, and so far indeterminable, chemical constitution of coal. With the new tools and the newer scientific knowledge available today, and with an adequate number of well trained workers, there is a real opportunity to secure facts regarding the nature of coal and its reactions that will be the foundation of further engineering advance in the use of coal.

More and more attention is being given to coal research. Since 1913 the Kaiser-Wilhelm Institute for Coal Investigations has maintained a group of men studying the chemistry of coal and its utilization. It must be recognized that in every field of utilization coal must undergo a series of chemical reactions to yield the result sought. These reactions must be understood to be properly controlled and so that the coal-handling equipment may be properly designed to give most efficient and economical operation. In 1917, the British Government established its Fuel Research Board. In 1930, Dr. Baker, president of the Carnegie Institute of Technology, organized the Coal Research Laboratory to do fundamental research on coal and its products. In other countries there are similar organizations, and in each country the contributions of numerous individuals in universities and industry are slowly leading to a better understanding of coal and its uses in industry. Progress is slow, in part because of the complexity of the problems and in part because of inadequate support of the work. An attempt is made in the following paragraphs to show where several industries, industry in general, and the public have direct interest in advances made in scientific knowledge of coal and of the processes of coal utilization.

The principal present and future uses of coal are combustion, carbonization and hydrogenation. In each of these coal is subjected to heat: in combustion, in the presence of air; in carbonization, in the absence of air; in hydrogenation, in the presence of high concentrations of hydrogen, a dispersing medium and a catalyst. A complete understanding of any of these processes must include, therefore, a knowledge of the reactions of coal caused only by raising its temperature, and fundamental research on any single process should contribute to a more efficient and economical use of coal in all. Scientific research on the mechanism of combustion, carbonization and hydrogenation should be encouraged by generous support of the coal industry—both mine owners and miners—the railroads and the Government, as well as industries that have special interest in only one of these processes. Most industries recognize that it is a good business principle to be able to inform their customers how to use their product best. This cannot be done by the coal industry because of lack of necessary information. The producers of competitive fuels, oil and gas, in this respect are in a much better position, which may account, in part, for the loss of the market for coal to these fuels. It would seem obvious that the coal industry should be most active in an attempt to secure the factual data needed to maintain its market for its product in uses for which it is most economical if correctly used, and to develop new methods of use that will permit it to compete in new markets. Organized mine labor should support research for the same reasons, so that the miner shall continue to have a job. Since transportation of coal contributes about 25 per cent of the gross revenue obtained by the railroads for the movement of all freight, the railroads should cooperate directly with the coal industry in support of coal research. The railroads have an added direct interest in that they use large amounts of coal. Governmental support of coal research should be much greater than in the past, because of the public interest, in conserving our less ample supplies of petroleum and natural gas for uses that cannot now be met by the use of coal, and in aiding industry in general to obtain its supply of energy most economically.

Let us consider separately the objectives of fundamental research in combustion, carbonization and hydrogenation, and point out the results that might be obtained from support of such work by industries that might profit from its successful conclusion.

Fundamental Combustion Research

Approximately 85 per cent of the coal mined in this country is burned in its "raw" state. The influence of coal quality, the specification of the fuel, and the design of the equipment are all empirical. A comprehensive program of combustion research should include studies of:

1. The specific requirements of temperature, volume of heat, uniformity of temperature, cleanliness, type of heat transfer, etc., for use of coal in domestic heating, and power and industrial heating.
2. The appliances available for burning coal for different purposes to determine their limiting characteristics with respect to completeness of combustion, effect on ash clinkering, effect of caking of coal, draft requirements, and refractory maintenance, and also to determine their convenience characteristics with respect to regulation, flexibility, and handling of fuel and ashes.
3. The effect of storage, of size stability, of handling and heating, and of grindability on the suitability of coals of different ranks and grades for use in different types of equipment.
4. Heat and fluid transfer in granular masses such as a fuel bed.
5. The mechanism of thermal decomposition of coal, since the action of heat on coal is an essential part of the combustion process and the method of its application may determine the furnace volume required for complete combustion of the liberated volatile matter and the combustion characteristics of the fuel on the grate.

6. The mechanism of combustion of the volatile matter liberated in the combustion process to make possible the most economical design of furnace for complete combustion.

7. The nature of the reaction between the solid fuel on the grates, or in suspension, and the oxidizing gas, and how this reaction is influenced by ash or added catalysts.

The first three studies are more in the nature of engineering research, which undoubtedly would point to the need of specific scientific data not now available. The last four subjects of investigation may be included in fundamental combustion research.

The ultimate aim of such a program is to secure data and methods of calculation by means of which the temperatures, gas compositions, percentages of coal and ash, and instantaneous combustion rates at any point in a furnace burning solid fuel may be ascertained, without actually building the furnace and operating it; and by which the effects of differences in fuel, gas velocity, method of firing, quantity of excess air and degree of air preheat can all be evaluated quantitatively for any given furnace design. This aim is still far from attainment, but it may be said that the ground work has been fairly well established over the last 10 or 12 years; the tools are at hand and the next 10 years should see a marked increase in usable data and methods of calculation.

To determine the possible benefits that may accrue from such a study of the combustion process, it is advisable to consider separately the five groups that would profit thereby: (1) the producer of heat and power by means of steam, and thus all manufacturing industries, whether they purchase or generate power and/or heat; (2) manufacturers of combustion equipment; (3) those engaged in the heat-treating of metals; (4) all those who operate domestic heating systems; and (5) coal producers.

The manufacturer of combustion equipment is the most obvious beneficiary of such a program. The following illustration shows what he stands to gain. A certain large public utility bought underfeed stokers for four boilers to supply steam for an 80,000-kw. turbogenerator at a cost in the neighborhood of $600,000. These stokers were rated at a much greater capacity and higher burning rates than any preceding ones; but, unfortunately, they failed to reach their guaranteed ratings by about 30 per cent. Here was a plain case of failure of design, due to inability to predict, on the basis of the empirical data available, how equipment similar to, but larger than, older equipment would operate —a failure that rational analysis of the problem could certainly have eliminated. For about three years, these stokers and their furnaces were tinkered with—new tuyere plates installed in this one, secondary rams replaced in that one, water walls covered up in a third, and later exposed again—partly at the expense of the operating company and partly that of the manufacturer. The manufacturers' engineers were in the plant almost continuously. Finally, the manufacturers developed, on other jobs, a complete departure from accepted practice in stoker construction, which did the work, and they tore out the old stokers and replaced them with the new design. Assuming that this was done in three years, and that 30 per cent of the sales price was held up by the purchaser until guarantees were met, the interest alone, lost to the manufacturer, would have paid for a research staff of two or three men for the same period, who might well have secured information that would insure against the recurrence of such a catastrophe.

One of the benefits that might accrue to producers of power is illustrated by the same story. In this particular case the utility had plenty of reserve capacity, so it was not actually forced to drop load; the loss entailed by the failure of the new equipment was due to the production of a portion of the firm load by old equipment at an increment heat rate in the neighborhood of 20,000 B.t.u. per kilowatt-hour, instead of by new equipment with an increment heat rate of 15,000 B.t.u. per kilowatt-hour. If this portion of the load amounted to 15,000 kw., at an average yearly load factor of 40 per cent, the loss from this source, assuming coal to cost $5 per ton, amounted to about $45,000 per year. Thus, if the utility and the manufacturer had previously supported a laboratory for combustion research, and had secured information making it possible to diagnose this case quickly and accurately, they would both have made a considerable saving on this one job alone. Although this particular instance involves stoker firing, it can scarcely be doubted that there have been similar experiences with pulverized-coal installations.

Another, even larger, source of benefit to all users of power equipment from the information to be derived from fundamental research on combustion is the possibility of determining for each particular installation the most economical size of boiler furnace. The economics of almost all the other items of equipment that go to make up a modern steam plant are susceptible of analysis, since the effects of variation in size and disposition of equipment can be evaluated; but up to the present there has been no way to make equally precise analyses with regard to furnace construction. In view of the large sizes and expensive types of construction in modern boiler furnaces, especially those for firing with pulverized fuel, it is imperative that means for making such an analysis be found.

There are many metallurgical heating operations where the presence of small amounts of ash would not be deleterious, but in which neither coal nor coke is used at present because of the difficulty, compared with oil or gas, of controlling the flame of solid fuel. Engineering research has failed to solve the problem; obviously, it must fail, since it cannot predict the phenomena occurring within the burning zone. Only fundamental scientific research on the mechanism of combustion can supply the information that is needed to determine how control of air velocities, preheat temperatures, and fuel characteristics can be manipulated to produce the qualities needed in the flame for proper metallurgical results. Since the cost of heat as coal is much less than that of oil or gas, and since the cost of heat may be a major part of the cost of producing metal articles, it is evident that the metallurgical industry may reap a large profit from the results of combustion research.

The low cost of heat in the form of coal makes it a very desirable fuel for domestic heating, but at present it has many disadvantages. It is smoky, dirty, and inconvenient to use in comparison with gas and oil. All of these difficulties could be eliminated, or minimized, by the use of properly designed small stokers. Many small stokers are now on the market, but none is completely satisfactory, and probably will not be until the mechanism of the combustion of coal on grates is better understood. Ash disposal and clinkering, in connection with both domestic stokers and power boiler furnaces, which furnish problems for engineering rather than scientific research, will not be satisfactorily disposed of until fundamental research provides complete knowledge of conditions within the combustion zone and thereby shows the conditions that must be met to obtain an ash that can be handled easily.

The interest of the coal producer in the study of the mechanism of combustion must now be clear. His only chance to regain the markets lost to gas and oil heating is by the perfection of small stokers or other devices for making domestic and small industrial heating plants clean, smoke-free, and trouble-proof. A new market to which the firing of solid fuels can be adapted is for the heating of metallurgical furnaces, which will be successful only when the temperature and gas composition can be exactly adjusted and controlled throughout the course of the flame.

In addition to these direct benefits, scientific investigation of the mechanism of combustion will secure other definite, indirect advantages. In the first place, the systematic knowledge resulting from this study will show the absolute limits that may be attained with regard to any phase of the combustion process, such as evaluating the benefits that may be expected from carrying out combustion at pressures above atmospheric; it will indicate in which directions progress may be expected, and where it would be uneconomical to search for improvement. Thus scientific research has a very important function as a control on engineering research. In addition, fundamental research on the mechanism of combustion requires additions to the sum of scientific knowledge, especially in the fields of fluid dynamics, heat transfer and the kinetics of heterogeneous reactions; and it may be expected that new processes, unforeseen at present, may be developed by its pursuit.

Summarizing, then, it appears that research on the mechanism of combustion of solid fuels aims at a complete understanding of physical and chemical conditions throughout the furnace; and that the tools are now available and the foundation prepared for securing this information, so that rapidly accelerated progress may be expected during the next 10 years. The results of this work will be of direct benefit; first, to manufacturers of combustion equipment and producers of heat and power, both public utility and industrial plants, since it will enable them to predict performance of combustion equipment accurately and thus to design economically; second, to operators of domestic heating equipment and to many manufacturers whose processes involve metallurgical heat-treating operations, to whom these results will make available an inexpensive fuel with the convenience and cleanliness of oil and gas; and, finally, to coal producers, for whom the results of this work offer a chance to regain lost markets and to open new ones.

Fundamental Carbonization Research

Approximately 15 per cent of the coal mined in the country is carbonized. Because of the scientific and engineering investigations that pointed to the advantages, primarily from the standpoint of fuel economy, of the byproduct oven over the older beehive oven, the carbonization process is somewhat better understood from the scientific viewpoint than is the combustion process. The understanding is still very incomplete, as the $50,000,000 lost by the public in low-temperature carbonization processes between 1920 and 1930 will clearly testify. This apparently simple change of carbonizing to 500° to 700° C., instead of the usual 900° to 1100° C., required modifications in the process and equipment for which inadequate scientific data were available to the engineer on which to base his design. Similarly, a review of the suggestion of Parr in 1926 to "condition" coal prior to carbonization at temperatures 25° to 50° C. below the initial softening temperature confirms the advantages to be anticipated from such a treatment, but also points out the many unknown factors that might change the gains expected to an over-all loss. The simple effect of completely drying the coal, and charging it to the oven hot, may so alter the yield and nature of the products that the usefulness of the process from the standpoint of increasing throughput and decreasing the trade waste, "liquor," may be questioned; the effects of the resulting change in the rate of heating on development of swelling pressures in the oven cannot be stated unequivocally. In spite of the development of numerous empirical tests of the suitability of a given coal for carbonization in a given type of oven, a full-scale oven test is still recognized as the only reliable criterion.

Excluding from consideration a study of carbonization equipment, a comprehensive program of fundamental research on carbonization is included in a study of the mechanism of thermal decomposition of coal and the effect of the original composition of the coal on this mechanism. A complete answer to this last phase, an understanding of the effect of the composition of the coal, includes not only a knowledge of the effects of mineral matter, moisture, or other inert material in the coal, but also a knowledge of the actual chemical constitution of the coal and how it may be modified to yield a satisfactory result in a given operation. From the results of investigation of the past 10 years, it appears that we are closely approaching the time when the essential chemical nature of a coal can be described fairly accurately. There will still be required many years of work before the action of heat on this complicated chemical material will be well understood. The separate effects of the maximum temperature and the rate of heating need much additional work. Such effects as may be due to density of packing, size distribution, moisture content, etc., on the course of the pyrolytic reactions need careful investigation in the laboratory as well as in the plant. Modifications in the nature of the coal substance by oxidation or reduction, and their effects on the subsequent carbonization at different final temperatures are fertile fields for study.

The ultimate aim of research on carbonization, which is far from realization, should be the ability:

1. To examine a given coal and predict with scientific accuracy the yields and nature of the products obtained in any given coke oven under any given operating condition.
2. To increase the range in choice of coals for the carbonization process.
3. To indicate the maximum monetary return from the products obtained.
4. To decrease the net cost of the process itself.

Even a partial realization of these aims should benefit: (1) the carbonization industry, by decrease in cost per unit production, in increase in markets for their products, and by possible simplification of their operations; (2) the steel industry, primarily in control of yield and properties of the coke produced for use in blast furnaces and in better markets for the byproducts; (3) the general public, by making available an inexpensive smokeless domestic fuel from high-volatile coals and reducing thereby the smoke nuisance, with its effects on health, cleanliness, and happiness; (4) the chemical industry, in providing it with a source of chemical raw materials of more uniform quality for the manufacture of dyestuffs, synthetic resins, pharmaceuticals, flavors, perfumes, etc.; (5) the electrical industry, in more uniform and cheaper insulating materials; and (6) the coal industry, in increasing the market for coal, since to make one ton of coke requires approximately 1.4 tons of coal.

The carbonization industry is dependent largely on two markets for its main product, coke: the steel industry for blast-furnace coke, and the domestic consumer for smokeless fuel. It seems unlikely that the first market will increase for some time above the capacity of existing installations. The second market undoubtedly will increase as the cost of a smokeless fuel, coke, decreases relative to that of coal and as stricter regulations are imposed by governmental action for the reduction of the smoke nuisance. Reduction in the cost of coke relative to coal may result from decreased cost of carbonization equipment, an increase in throughput of existing equipment, and/or an increase in value of the byproducts obtained. A complete understanding of the carbonization process would quite certainly bring about the first two. For example, if a satisfactory method of pretreating coal to remove all free moisture, and part of the water of decomposition, at temperatures not far below the initial softening temperature, and of carbonizing the heated coal, were developed, the capacity of existing coke ovens should be increased about one-third with an expenditure of from 10 to 25 per cent of that required by a similar increase in number of coke ovens. The increase in value of byproducts will come only with a better understanding of what they really are and how they may be refined into fractions having closely reproducible and known chemical properties, so that it may become more profitable to refine the tar produced than to burn it as fuel.

One frequently hears that it is a "crime" to burn raw coal because of the "valuable" byproducts contained therein, which might be obtained by carbonization. It is not realized that in an average year about half of the tar produced at byproduct ovens is burned without refining. This is due, in part, to lack of knowledge of the nature of the chemical constituents in the tar and how to obtain them in the requisite degree of purity for possible chemical markets, as is indicated by the fact that in the 10 years from 1923 to 1933 coal-tar finished products increased only 46 per cent, coal-tar intermediates only 61 per cent, while synthetic organic chemicals not from coal tar increased 760 per cent. The first commercial synthetic resin, which has had such wide use as an insulating material in the electrical industry, was produced by condensation of coal-tar phenols with formaldehyde, but these phenolic resins are in many instances losing out in the competition with some of the newer resins not derived from coal tar, owing, in part, to variability of the coal-tar phenolic fractions. Study of these fractions should indicate changes in the methods of refining to give a satisfactory product and maintain for the phenol resins their predominant position in the synthetic resin market. An anomaly in this connection is the competition of synthetic phenol with byproduct phenol: the chemical industry can purchase benzol—a byproduct of coal carbonization—and put it through a series of chemical reactions and still compete on a price basis with phenol obtained directly as another byproduct of coal carbonization.

One of the most important byproducts of carbonization is gas. Adoption of any modification of the carbonization process would depend, to a large extent, on its possible effects on the yield and calorific value of the gas obtained. On the other hand, deliberate changes in the carbonization process to obtain, from a given coal, gas of known composition must be based on knowledge not now available, but which should be obtained from fundamental scientific research on the action of heat on coal. Recent laboratory studies have shown that with a given coal a change in the rate of heating does not affect the yield of gas as per cent by weight of the coal coked, but does markedly affect its composition and calorific value. Direct gasification of coal or coke, as in the production of water gas or producer gas, is more directly related to combustion than to carbonization, and scientific research on the mechanism of combustion should show how to control better the reactions involved. In gasification the oxidizing gas is steam or carbon dioxide while in combustion the oxidizing gas is oxygen from the air and steam and carbon dioxide from the reaction of the oxygen with the coal, or its volatile matter.

These illustrations of the benefits that may be expected to result from scientific research on the carbonization of coal are typical only, and should not be regarded as exclusive of many others. Realization of the objectives outlined would, however, well repay a considerable investment in fundamental carbonization research to the contributing industries, which should include the coal producer, the coke-oven builder and operator, the general public, the steel industry, the chemical and electrical industries, and the gas manufacturer. Immediate returns from such an investment might not be large, but the results obtained in a program sustained for a continuing period of years should be increasingly large, as experience in scientific research in other industries shows.

Fundamental Hydrogenation Research

The proven oil reserves of the United States are adequate for a period of only about 15 years. The rates of production maintained during the past several years have been possible only by discoveries of new pools with their flush production. Predictions regarding the time when an oil shortage will be felt industrially in this country are necessarily based on estimates of new discoveries and therefore are a matter of opinion; they range from 5 to 8 years hence to the time of the "next generation." On the other hand, there is no disagreement that our coal reserves are adequate for any anticipated demand for many centuries and that as oil production decreases the liquid fuels required by industry and the public will be obtained from coal in increasing amounts.

Hydrogenation of coal, and also of coal tar, to produce motor fuel is carried out in England and Germany on a large scale. The process is expensive, high pressures and temperatures being necessary, and at present cannot be operated at a profit without Government subsidy. The economic situation will change undoubtedly as more experience is obtained in the operation, as more knowledge is obtained about the chemical nature of coal and about the reactions involved, and as petroleum reserves become depleted. Mr. Farish, of the Standard Oil Company of New Jersey, made the statement in 1934 before the House Committee on Petroleum Investigation that gasoline could be made with existing processes at 12¢ per gallon from coal on a basis comparable to 5¢ per gallon from petroleum. Undoubtedly we shall profit in this country from the experience of the British and Germans obtained in plant-scale, expensive experiments. It would seem wise to be forehanded in this matter and obtain as much knowledge as possible about the chemical nature of American coals and their suitability as a raw material for production of liquid fuel before industrial requirements demand coal hydrogenation in this country.

Fundamental coal hydrogenation research should benefit directly: (1) the coal industry, which may look to a new market approaching with time a magnitude not much less than the present total production; (2) the oil industry, which will, in the future, need to supplement crude petroleum to supply the demand for refined products, and (3) the general public and all who are dependent on liquid fuel for transportation or other purposes for which a solid fuel is not adapted.

A comprehensive program of fundamental research may be outlined by a generalized description of the present process. The coal of unknown chemical constitution is suspended in a liquid medium, which exerts an unknown solvent, or dispersing, action on it; the mixture is then treated with hydrogen, which reacts in an unknown way, at an elevated temperature, which itself has an unknown effect on the coal used. The products obtained are of unknown chemical constitution; the reaction is carried out in the presence of catalysts whose specific effects are still unknown. It has been stated that one-third of the hydrogen used reacts to give waste products—water, ammonia and hydrogen sulfide—by reaction with the oxygen, nitrogen and sulfur in the coal, one-third of the hydrogen used reacts to give gaseous hydrocarbons, and only one-third of the hydrogen is obtained in the liquid products of the reaction for which the process is primarily operated. It would seem a profitable field for research to study cheaper methods of pretreatment of the coal to increase the over-all efficiency of the hydrogenation reaction to produce liquid fuel. Also, if it were found possible to hydrogenate coals directly with steam to yield liquid fuel and carbon dioxide, the economy of the process should be much improved. In the present process steam is the source of hydrogen, which is produced in a separate step by reaction with carbonaceous materials to yield hydrogen and carbon dioxide. From 1914, the time of the first Bergius patent on coal hydrogenation, the progress has been great, although made largely through empirical, but systematic, methods. Further progress should be made as our understanding of the nature of the process itself increases.

In both carbonization and hydrogenation research, study of the chemical constitution of coal assumes greater importance than in combustion research. One byproduct expected from such a study is an increase in the use of coal as a raw material for the chemical industry.

A continuing program of research on the nature of the reactions occurring in coal hydrogenation should receive the support of the coal, oil and chemical industries and the general public, all of whom should be benefited by the progress made before commercial development of the process is a necessity from the standpoint of national economy.

CONCLUSION

Some general, and some more specific, examples have been given to show how fundamental research on combustion, carbonization and hydrogenation should benefit the coal, railroad, metallurgical, power, oil, gas, chemical, and electrical industries and the general public. It is difficult to estimate the support now given to such work in the United States but, judging from the amount of work outlined that remains to be done and has not yet been systematically attacked as reported in the literature, the support is much less than would seem to be warranted by the results that may be expected. The coal bill of the country is probably more than one and one-half billion dollars a year. If fundamental research brought about an increase of only one per cent in the national fuel economy, which is an absurdly low estimate, the saving to the public would amount to more than fifteen million dollars a year. It would not seem unreasonable to spend a million dollars a year on a well planned continuing program in which the best scientific talent should be employed. However, there is at present no organization in this country that has the breadth of view and experience in scientific research needed to expend such a sum wisely. If such a sum could be obtained for scientific research on coal, it should certainly be administered by an advisory board composed of men with wide experience in directing scientific research for industry, and it seems appropriate that such a board should function under the auspices of the National Research Council. The only apparent practical method of stimulating coal research in this country immediately would be to support and encourage the relatively small number of groups now at work as nuclei of a broader and more scientific program.

DISCUSSION

(*John T. Crawford presiding*)

J. E. TOBEY,* Cincinnati, Ohio (written discussion).—The writer concurs completely with Dr. Lowry in his treatment of this most timely subject. The coal industry as a whole has been remiss in its attitude towards research and as an industry it perhaps knows less about its product and its proper utilization than any other major industry. Such a situation is conducive to economic death.

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* Manager, Fuel Engineering Division, Appalachian Coals, Inc.

Owing in a large measure to the lack of fundamental research, coal is badly misunderstood and is looked upon by many as a simple product of which the proper utilization is rather easily accomplished, but as a matter of fact coal is one of the most complex organic compounds with which man has to deal and its satisfactory utilization is difficult to attain.

Referring to fundamental combustion research, fuel engineers who are engaged in the application of coals to various burning equipment are greatly handicapped by the lack of basic information as to the combustion process itself, particularly the mechanics of combustion, and how different coals react to that process. Manufacturers of combustion equipment suffer likewise. Equipment cannot be designed to properly burn coals until fundamental research discovers the underlying principles governing the combustion of complex fuels.

To emphasize this please consider the following:

1. No one at present can explain specifically what occurs in a stoker fuel bed.
2. Little is known of the plastic properties of various coals and their effect upon combustion behavior.
3. Why coals considered to be similar give dissimilar combustion results.

The solution of these and many other combustion problems are dependent on fundamental research.

The relationship between the scientist and the engineer can be likened to an electrical circuit in which there is a generator and a motor; one cannot function without the other, they operate in a closed cycle.

Those whose interests lie in coal should recognize the vital part that research plays in its successful exploitation and give it the aggressive support that is so necessary. Research is coal's best ally.

A. R. Powell,* Pittsburgh, Pa. (written discussion).—I believe that it was Arthur D. Little who said that there are three kinds of research: pure research, applied research and misapplied research. Fortunately, the last named variety is not possible when an honest and useful objective is sought and when care and intelligence is used in the planning and prosecution of the experimental work.

C. E. K. Mees says: "The difference between a pure and an applied science is therefore merely one of intention. An investigator in pure science desires primarily to advance knowledge, while an investigator in applied science desires to obtain a financial advantage from his work, but the purely scientific work may eventually prove of great financial value while that of the industrial investigator may have much value for the theory of the subject."

I feel sure that the type of research outlined by Dr. Lowry provides a desirable linkage between pure science and applied science as related to the use of coal. The intention is to advance our fundamental knowledge of coal and what happens to it when it is burned or processed, and the method of attack involves the use of the most modern knowledge and tools that have been developed by so-called pure science. By thus evaluating these fundamentals of the properties and changes undergone by the coal, we come closer and closer to the ideal state where we can predict by calculation just what will happen under a given set of conditions instead of the necessarily superficial tests by trial and error now in use for most of the applied research on coal.

Unfortunately, the usefulness of the results obtained from this, let us say, fundamental applied research, and their broad application to industrial processes are not so generally recognized as they should be. We continually see examples, in many of our universities and other scientific institutions, of research problems having no broadly applicable objective, often being merely repetitions of previous superficial studies of the same problem, and having only the rather dubious advantage of providing an argument for the appropriations of funds by throwing a theatrical glamor around a project that is useless, despite publicity claims of huge savings or the discovery of new sources of wealth. Usually this type of misapplied research is frowned on by the research director himself, and he would like to be delivered from it. Those of us in industry who have some connection with problems involved in the use of coal can do much to correct this matter by encouraging in every possible way the collection of useful fundamental data on coal by institutions with personnel and equipment adapted to such work. Often the industries themselves cannot carry such projects in their own research groups, but they can secure this valuable assistance by properly encouraging institutions that are properly equipped for fundamental research.

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* Research Division, The Koppers Construction Co.

Dr. Lowry has drawn a picture with commendable clarity of the scope of future fields of research on coal-utilization problems. Let us use our best efforts to see that these problems are carried through in a scientific manner!

A. C. Fieldner,* Washington, D.C. (written discussion†).—Dr. Lowry has presented a sound and comprehensive case for coal research. Discussion can only amplify some of his arguments and provide additional illustrations of the value of coal research. We must agree with him in the stress he has placed on the need of thoroughly scientific fundamental work on the nature of coal and on the reactions that take place in its combustion, carbonization or other processing. Knowledge of these fundamentals will go far toward eliminating the time-consuming empirical experimentation that industry now follows in developing processes for the better utilization of coal.

Much of the fifty million dollars expended on large-scale low-temperature carbonization plants that failed could have been saved if a fraction of this amount had been devoted to finding out the relationships between the chemical and physical composition of different coals and the effect of heat-treatment on their plastic and caking properties.

We know very little of the nature of the reactions that take place on conditioning coal with heat or air before coking it at low or high temperatures. Such conditioning is a necessary part of certain processes for the production of smokeless fuel. In the absence of fundamental knowledge an expensive series of large-scale tests with each coal, or blend of coals, is the only certain means of working out the proper plant operation.

Sir Frank Smith, Secretary of the Department of Scientific and Industrial Research of Great Britain, in a recent address on Coal, Power and Smoke, called attention to the fact that "we do not know the structure of coal and are correspondingly handicapped in its treatment." Recognizing this need, the Fuel Research Board of Great Britain has conducted for some years a broad research program at the Fuel Research Station. An important part of this program has been a survey of the nation's coal beds to obtain the fullest possible information on the chemical and physical properties of British coals as they occur in the ground and after suitable cleaning and preparation for industrial or domestic use.

In the United States the Bureau of Mines, the U.S. Geological Survey and certain state surveys have made some progress in determining and publishing proximate and ultimate chemical analyses, calorific values, and ash-fusibility tests of some of our more important coal beds. Recently the Bureau of Mines, in cooperation with the American Gas Association, has developed a procedure for surveying the gas-making, coke-making and byproduct-making properties of American coals. Work on the survey itself is now under way. The results thus far obtained have shown the inadequacy of many of the empirical tests proposed from time to time as measures of the coking properties of coal. Much additional fundamental research is needed to determine the mechanism of softening and swelling of coal under the action of heat before the softening behavior of any coal can be predicted accurately from simple laboratory tests or from a chemical, physical, or microscopic analysis of the coal.

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* Chief Engineer, Experiment Stations Division, U.S. Bureau of Mines.

† Published by permission of the Director, U.S. Bureau of Mines.

Lack of knowledge of the constitution of coal was a severe handicap to the Sectional Committee of the American Society for Testing Materials in developing standard specifications for the classification of American coals. In fact, no specifications have yet been developed for classification by type solely because the Committee does not know enough about the relation between origin, microstructure and constitution of the different kinds of coal to develop sound specifications.

Dr. Lowry has called attention to the importance of knowledge of the constitution of American coals in connection with their utilization for the production of liquid fuels to supplement an inadequate future supply of petroleum. No matter when our petroleum production may begin to diminish, it is none too early for systematic research on substitutes for coal. The problem is a difficult one.

The need of more coal research is so great that we need not concern ourselves about duplication. Research is one field of endeavor in which there should be no fear of overproduction. Of course I mean research by competent investigators, properly directed in well equipped laboratories. Industry should give these laboratories continuous support, so that they can carry out the necessary long-range research program required for solving the difficult problems underlying the better utilization of coal.

R. H. Sweetser,* New York, N.Y.—One phase of this discussion interests me; that is, the burning out of iron in the grates. At a meeting in New York last month, I gave a lecture on Blast Furnace Construction and Operation, and made the statement that we do not know just what takes place inside the upper part of the blast furnace. After the lecture one young fellow asked: "What difference does it make what takes place there?"

I said, "All the blast-furnace construction and operation that has been carried on for many years is based on one theory or another, but most of those theories are wrong, and until we find out what really takes place inside of the shaft of the furnace we will continue to have expensive construction and operation."

Here is a place in which we are using coal and we really do not know the fundamentals of the combustion of the coal, or, for that matter, of whatever the fuel is, and for that reason more of this fundamental research should be carried on. In blast-furnace work we have long felt the lack of such knowledge but have not yet succeeded in overcoming our difficulties. Therefore I think this paper by Dr. Lowry should receive great attention. He said, himself, that by spending a million dollars in research we might save fifteen million dollars. If that is so it may mean that coal consumption is going to be cut down that much. It reminds me of the campaign I started about fifteen years ago for getting the ash out of coking coals, and that showed, as a result, a considerable reduction in the amount of coke used per ton of pig iron. There was a great decrease in the amount of coal that had to be mined. But the research will do other things besides reducing the amount of coal now used in metallurgical operations. It can find further uses for coal, and the cheaper and the more conveniently we develop uses for coal the more demand there will be for it.

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* Consulting Engineer.

Mechanism of Combustion of Coal

By Martin A. Mayers,* Pittsburgh, Pa.

(Pennsylvania State College Meeting, October, 1934)

Five-sixths of all the coal that is mined in the United States is burned, without previous treatment other than screening, for the production of heat and power, so that its value is fixed by its suitability for this purpose. Nevertheless, the characteristics of individual coals that determine their fitness for use in particular pieces of combustion equipment are not yet well known. This condition is due not only to the small extent of our knowledge concerning coal itself, but also to our comparative ignorance of the nature of the combustion process. The following paper is an analysis of the combustion process as it occurs in industrial furnaces, based largely on the results of laboratory research on combustion rates. From this analysis it is possible to determine the characteristics of the fuel that limit the capacity of a given piece of combustion equipment. Limitations imposed by other factors than the attainable speed of combustion will not be considered, since they are subject to a different type of analysis involving the balancing of maintenance costs against fixed charges.

In order to observe the relations among the various factors that affect the speed of combustion, representative particles of fuel will be followed through two types of furnaces. A pulverized-coal furnace will be considered first, because the independence of the individual particles facilitates the analysis of the effect of furnace conditions. The second example will be a grate-fired furnace in which ignition of the fuel takes place on the underfeed principle. This includes both underfeed and chain grate stokers,(8)† but does not include hand-fired furnaces and some overfeed stokers. The furnace will be considered to be large enough, the temperature high enough, and enough air provided, to burn the fuel completely, without, for the moment, specifying these quantities more completely.

FURNACES BURNING PULVERIZED COAL

In a pulverized-coal furnace, the fuel particle is carried, in a stream of air, into the furnace, where it is subjected to radiation from the flame and from the hot walls of the furnace. The increase of particle temperature due to its exposure to radiation may be calculated to give results similar to Fig. 1. This calculation was first given by Nüsselt⁽¹⁾ but, because of his choice of assumptions, he obtained the anomalous result that the maximum temperature attained varied in proportion to the radius of the particle. The calculation has been repeated here using assumptions that free the result of this limitation. The coal heats up very rapidly, the time from its entrance to the furnace until it reaches about 350° C. being only 60 milliseconds for the conditions of this figure.

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* Coal Research Laboratory, Carnegie Institute of Technology.

† Superior figures in parentheses refer to the list of references at the end of the paper.

At about this temperature the coal particle begins to give off its volatile constituents. There is some doubt about what occurs at this point. The temperature is well below that of ignition for the gases given off by the coal, but experiments on coal in bulk indicate an ignition temperature in this range. Probably what occurs lies between two extreme conditions, depending on the type of fuel. When the volatile matter contains large quantities of heavy hydrocarbons, these may be ignited by virtue of their comparatively low ignition temperatures and also because of some slight catalytic activity of the solid fuel surface. When this occurs, each individual particle acts like a miniature gas jet, having a tail of gas flame extending ahead of it. This flame helps to heat the coal to the point where the rate of combustion of the solid residue becomes appreciable. The other extreme condition occurs with coals whose volatile matter contains mostly methane and other simple gases of which the ignition points are relatively high. In this case, the volatile matter passes off into the gas stream without being ignited, and is not burned until the gas reaches a region of higher temperature somewhat farther along in its path; the volatile matter plays no part in the ignition of the fuel particle except as it increases the flame temperature.

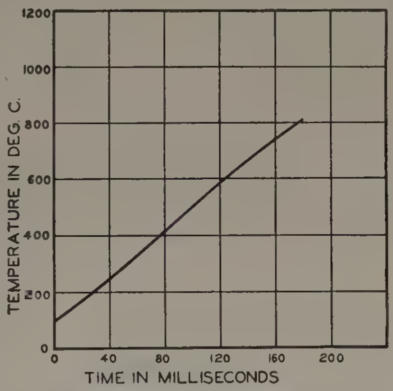

FIG. 1.—TEMPERATURE OF A COAL PARTICLE EXPOSED TO RADIATION.
Particle diameter 50 μ; furnace temperature 1500° C.; initial temperature of particle 100° C.

Let us consider the second case in detail because it is more easily subject to calculation, and because it represents the worst condition; i.e., that of slowest ignition. From the equations for heat transfer to a suspended particle it is apparent that the quantity of heat absorbed in raising the temperature of the particle is negligible by comparison with the quantities received by radiation, and lost to the surrounding air by convection. For this reason the existence of endothermic or exothermic reactions during the distillation process has no effect on the rate of increase of temperature. Hence the particle will continue to be heated by radiation until the reaction of the oxygen in the atmosphere with the fuel, which is now coked, becomes appreciable. The point at which this occurs, somewhere in the neighborhood of 500° C., depends on the reactivity of the coke formed. The chemical reaction is not yet well understood,(2) but it is known that its rate varies with temperature according to an exponential term of the form

$$e^{-\frac{E}{RT}}$$

a quantity which, in a certain region, increases extremely rapidly with temperature. Hence, as soon as the reaction becomes great enough to raise the temperature, a self-perpetuating process is started, which would cause the particle to reach an extremely high temperature and to burn almost instantaneously if it were not for the exhaustion of the oxygen in the atmosphere immediately surrounding the particle.

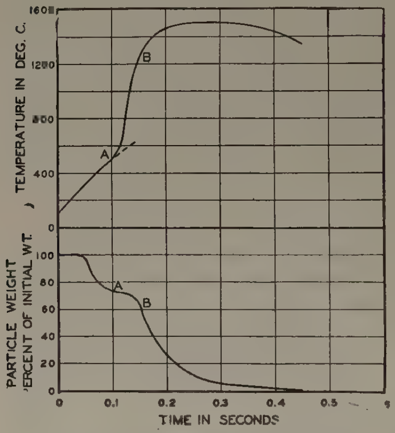

FIG. 2.—PROGRESS OF COMBUSTION OF A SUSPENDED COAL PARTICLE.
A-B represents ignition zone.

The stage now reached, represented by point *B*, Fig. 2, has occupied only a fraction of a second. The particle has lost little weight in addition to its volatile matter, but it has almost reached the flame temperature and the conditions have been met for the last and longest part of the process, the combustion of the solid residue.

The principles that control this process are well understood. It is apparent that at this stage of the combustion process, a large proportion of the molecules of oxygen that impinge on the hot fuel surface immediately burn to carbon dioxide or carbon monoxide. However, there is only a limited amount of oxygen in the immediate vicinity of the particle and this is quickly exhausted. When it is considered that a particle of fuel requires about 60,000* times its own volume of air to burn it, it is evident that a large part of the air must come from comparatively great distances. This transport of air is brought about partly by diffusion of oxygen along the concentration gradients set up by the rapid use of the oxygen in the immediate neighborhood of the particle, and partly by the fluid flow of air about the particle due to the relative velocity between the particle and the air caused by their difference in density. Under these conditions the rate of burning is determined entirely by the rate of transport of oxygen, not by the speed of the chemical reaction.

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*Originally printed 1000, by mistake.

These principles can easily be stated as differential equations,(2) but the solution of these equations has not yet been found. However, there are available some experimental determinations(4,5,6) of the rate of combustion. In addition, several workers(1,2,3,4) have obtained, by the use of various simplifying assumptions, approximate solutions to the differential equations, which, by adjustment of various constants, can be made to represent the experimental results. By correlating these data it may be concluded that the rate of combustion of a suspended particle:

1. Varies approximately as the radius of the particle. It can be shown that when this is the case, the time of complete combustion is proportional to the square of the initial radius of the particle, and that the time of burning down to any radius varies as the difference between the squares of the initial and the final radii. This permits a comparison with the experimental data of Smith and Gudmundsen,(5) from which Fig. 3 was drawn. It is apparent that the data agree closely with the straight-line relation between time of combustion and the difference between the squares of the initial and final radii, predicted by theory.

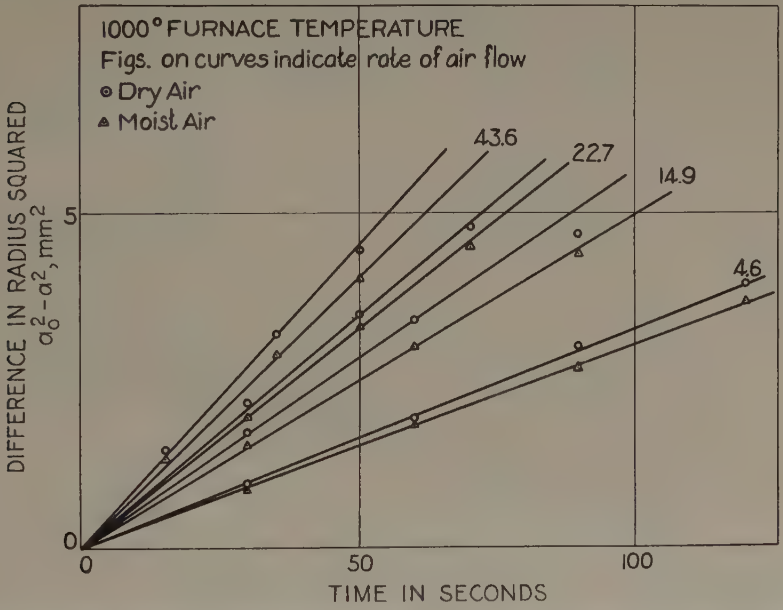

FIG. 3.—COMBUSTION OF GRAPHITE SPHERES. *(From data of Smith and Gudmundsen.(5))*

2. Varies approximately as the 0.4 power of the relative velocity between the particle and the atmosphere. This result was obtained(3,4) by using the concept of a stagnant film about the particle whose thickness decreases with increased gas velocity, an approximation extensively used in heat-transfer calculations. The experimental data of Smith and Gudmundsen(5) and those of Tu(4) for high temperatures support

this conclusion, for when their combustion rates are plotted against velocity to logarithmic scales, as in Figs. 4 and 5, the slope of the straight lines obtained is 0.4. The value of the exponent of the velocity, 0.6, obtained by Rosin(10) from model tests on the solution of salt cubes is apparently too high.

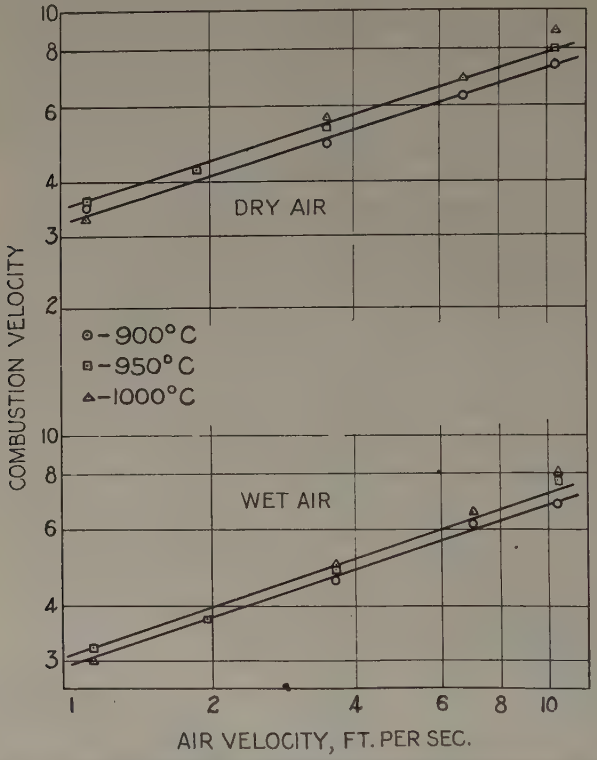

FIG. 4.—EFFECT OF GAS VELOCITY ON COMBUSTION RATE OF GRAPHITE SPHERES. (*From data of Smith and Gudmundsen.*(5))

The variation of the combustion rate with temperature has not yet been satisfactorily determined. Tu(4) and Smith and Gudmundsen(5) found the combustion rate to vary as some power of the temperature between 0.5 and 1.5 when the relative velocity between the air and the particle did not change, while Griffin, Adams, and Smith(6) found that with suspended particles the temperature coefficient of the combustion rate was negative. While this observation may be correct, owing to a negative temperature coefficient of velocity of a suspended particle, it still requires confirmation.

The results enumerated above were obtained with unlimited air supply to the particle. With finite quantities of air, the time required for complete combustion of a particle varies as shown in Fig. 6, where the ratio of the time required with any percentage of excess air to that with an infinite amount of air is plotted against excess air(1,7).

From these results, it may be concluded that, from point *B* of Fig. 2, the combustion process follows the course shown. The weight of the particle follows a hyperbolic curve similar to that indicated in the lower part of the figure, the slope of which is determined by the relative velocity of the particle with respect to the atmosphere, the initial size of the particle, the amount of excess air, and to a lesser extent, the temperature of the combustion space.

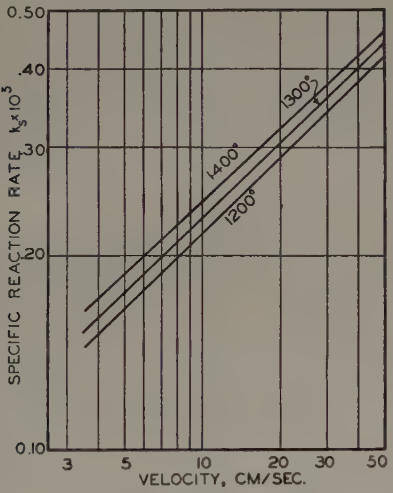

FIG. 5.

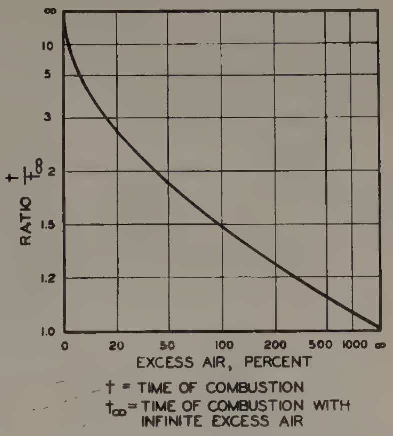

FIG. 6.

FIG. 5.—EFFECT OF GAS VELOCITY ON COMBUSTION RATE OF GRAPHITE SPHERES. (*From data of Tu, Davis, and Hottell.*(4))

FIG. 6.—EFFECT OF EXCESS AIR ON TIME REQUIRED FOR COMBUSTION.

The specification of furnace temperature and of the required combustion volume referred to above can now be more nearly completed. The required furnace temperature is fixed by the time available for ignition, and may be calculated by means of the principles discussed previously. It requires a knowledge of the ignition temperature of the coal and the size of the particles fired. The furnace volume is determined by the total time required for combustion and by the velocity of the gases through the furnace. Thus high linear velocity of gas flow, which is desirable because the turbulence associated with it promotes effective mixing of the gases, is undesirable because it decreases the time during which the burning particles are in the furnace. If, however, the kinematics of the gas flow can be controlled so that a high linear velocity is obtained with low net velocity through the furnace, both of these features may be secured, while an additional advantage, that of increasing the combustion speed by an increase of the relative velocity of the fuel particles with respect to the air, may also be obtained. This is the threefold advantage secured by furnaces that are so fired as to maintain a cylindrical vortex in the combustion space, such, for instance, as the well-type furnaces of the Buffalo General Electric Co. or the Combustion Engineering Corporation "steam generator." Because of the long, nearly horizontal part of the end of the curve of Fig. 2, which implies that a large part of the furnace volume is used to burn out the last fraction of combustible, it may be economically desirable to design furnaces for considerably less than complete combustion. The saving in investment costs due to the decreased furnace size may more than pay for the fuel lost as combustible in the refuse.

Grate-fired Furnaces

In a grate-fired furnace the analysis is complicated by the interdependence of the particles within the bed, and by the fact that the thermal properties of the coal itself become of considerable importance. The process has not yet been thoroughly analyzed, but it is possible to find the same three stages here as in the firing of pulverized fuel. Thus, in the diagrams of Fig. 7, the heating up stage occurs in the region below and to the left of the heavy dashed line; ignition occurs at this line; and burning, above and to the right. There are, however, several important differences between the mechanisms in grate firing and in pulverized-fuel firing.

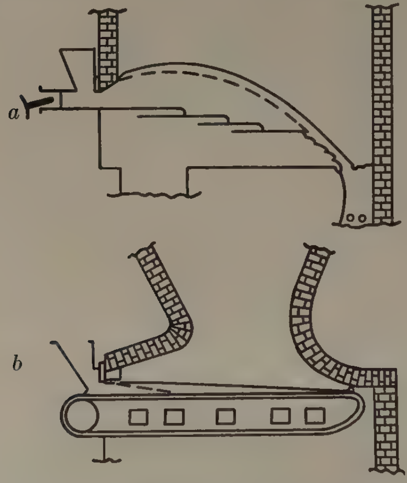

Fig. 7.

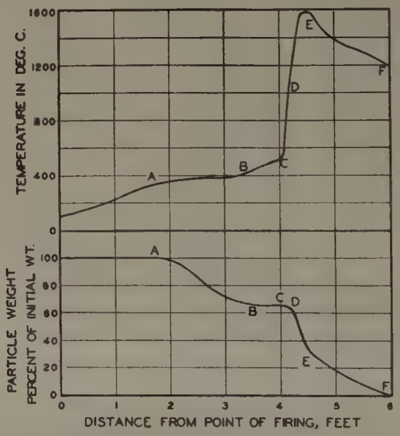

Fig. 8.

Fig. 7.—Diagrammatic cross-sections of stoker fuel beds.
Dashed line represents ignition zone; the region below and to left, the heating zone; that above and to right, the zone of active burning.

Fig. 8.—Progress of combustion of a representative particle in a fuel bed.
Length of stoker about 15 ft.; *AB* represents evolution of volatile matter; *CD*, ignition; *DE*, active burning; *EF*, reduction of carbon dioxide.

Because of the fixed position of critical zones in a stable fuel bed, it is more convenient to refer to the speeds of the various phenomena in terms of space intervals, instead of the time intervals to which they are equivalent. It is evident that, in a stable fuel bed, the rate of advance of the ignition zone and of the end of the burning zone—the top surface of the fuel bed—is exactly as great as the rate of coal feed, in the opposite direction. Plotting the temperature and weight of a particle against distance from the point of firing for a representative particle, curves similar to Fig. 8 are obtained. Here the region *C-D*, representing ignition, corresponds to the segment *A-B* in Fig. 2, and occurs at the heavy dashed line of Fig. 7.

The heating curve for grate firing is considerably more complicated than for pulverized-coal firing. It also represents a much longer period of time, estimated at ½ hr. for an underfeed stoker at high combustion rates, and several minutes on a chain grate. The temperature at any point is determined by a balance between the heat gained at that point by conduction and convection from hotter parts of the bed and by exothermic reactions which may take place there, and that lost by conduction to cooler parts of the bed, to the primary air blast, or to endothermic reactions. The term "conduction" as used here has a purely formal meaning; the treatment of the flow of heat in a bed of crushed fuel is similar mathematically to that of metallic conduction, although the mechanism of heat transfer between adjacent particles is probably radiation. The rate of heating at any point depends on the equivalent conductivity of the fuel bed at that point and on the temperature gradient, which is set by the geometry of the fuel bed. The heating process is slow because of the low thermal conductivity of the fuel bed and the fact that the temperature gradient must be small at the grate surface to avoid burning the stoker parts. Because the amount of heat conducted is comparatively small, the heat capacity of the coal itself is no longer negligible, as it was for pulverized fuel. Hence the existence of endothermic reactions during the process of distilling off volatile elements will affect the temperature distribution, causing a flattening of the curve of temperature, at *A* in Fig. 8, in the region of distillation, with an abnormally rapid rise immediately beyond it at *B*, due to the high conductivity of coke as compared with coal.

As with pulverized fuel, this is followed, at some point depending on the reactivity of the coke, by ignition, *C-D* in Fig. 8, and active combustion. In this case, however, the ignition zone is very much shortened because of the immediate proximity of the zone of active combustion, so that it amounts to little more than a line of demarcation between the zones of heating and of active burning. The region of active combustion differs from the corresponding region for pulverized coal by effects due to the differences in the geometry of the two systems, and by effects due to the high relative velocities that may be obtained between fuel and air. Instead of the suspended sphere, subject to diffusion from all directions, the fuel bed must be thought of as a collection of small channels, as shown in Fig. 9, running from the ignition zone to the free top surface of the bed, through which the primary air flows. Then the rate of combustion will no longer depend on the size of the fuel particles, but primarily on the velocity of the air, and also on the dimensions of the channels and the temperature. The air velocity relative to the fuel may be so great that the "diffusion resistance" may be decreased to as little as one-twentieth of that associated with pulverized-coal firing. If, as is usual on underfeed stokers and at the head of chain grates, the channels are long enough, or in other words, if the fuel bed is thick enough above the zone of ignition, all the oxygen in the primary air will be completely consumed in only a fraction of the length of the channel. The carbon dioxide thus produced will then be further reduced to carbon monoxide, since it is still in contact with carbon, but at a considerably lower rate.

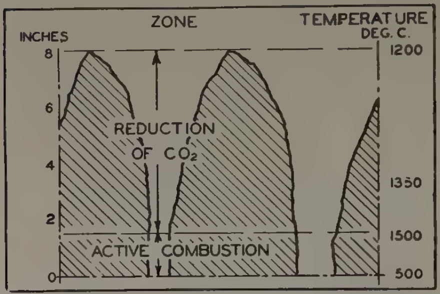

Fig 9.—Air passages in stoker fuel bed. Plane of ignition at distance zero.

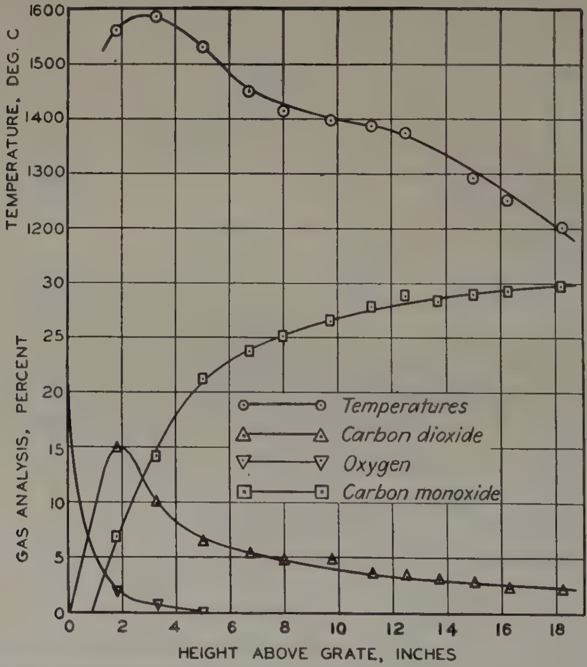

Fig. 10.—Temperatures and gas analyses in an overfeed fuel bed of high-temperature coke. (*From paper by Nicholls and Eilers.*(8))

Hence the final stage of the consumption of the fuel is characterized by action similar to that of a gas producer.

Since all the oxygen is consumed in passing through the fuel bed, the rate of burning is proportional to the rate of air flow through the bed. It was shown earlier in the paper that the rate of combustion of pulverized fuel per unit surface varies as the 0.4 power of the velocity; in grate firing, in the presence of an excess of coke, the total rate of burning varies as the first power of the velocity. Hence, the extent of the zone of active combustion, or the length of that part of the channels shown in Fig. 9 devoted to active combustion, varies as $\frac{V}{V^{0.4}}$, or as $V^{0.6}$. As a mixture of combustible gases rise from the fuel bed, owing to the producer action of the bed, it is necessary to add secondary air, either over the fire, or as leakage, to burn these gases, and to provide sufficient furnace volume to insure their combustion.

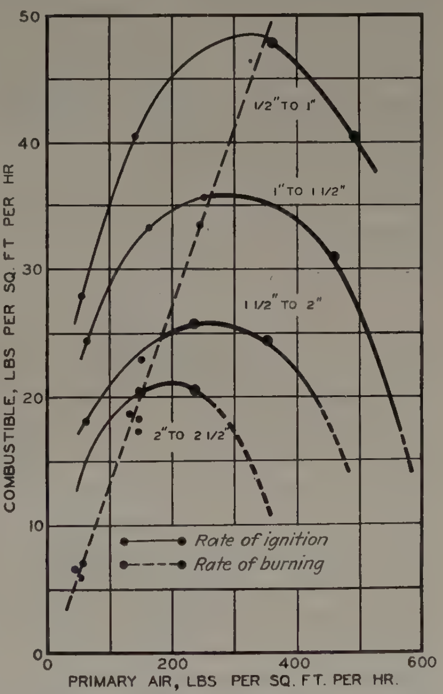

FIG. 11.—RATES OF IGNITION AND OF BURNING OF HIGH-TEMPERATURE COKE. (*From Nicholls and Eilers*⁽⁸⁾.)
Figures on curves indicate size of coke.

This concept of the phenomena of grate firing is well substantiated by the recently published experimental work of Nicholls and Eilers.⁽⁸⁾ This, with the earlier work of the Bureau of Mines,⁽⁹⁾ appears to be the only available experimental investigation of the characteristics of grate firing. The model tests of Rosin⁽¹⁰⁾ on the rate of solution of beds of salt serve only to demonstrate the importance of diffusion as a factor controlling the speed of reaction, but otherwise give little information. Several curves taken from the paper of Nicholls and Eilers are shown here as Figs. 10 and 11. Fig. 10 shows temperatures and gas analyses at various heights in an overfeed fuel bed. This corresponds to conditions from the beginning of the ignition zone through the region of active combustion of Fig. 8. The form of the temperature curve is similar to that of Fig. 8, showing a sharp rise through the ignition zone and the first part of the combustion zone. The gas analyses show that this rise corresponds to the complete reduction of the oxygen in the air to carbon dioxide. Beyond this point, the temperature falls rapidly, because, as the gas analysis shows, the carbon dioxide is being reduced, by an endothermic reaction, to carbon monoxide. This reaction is virtually complete at a distance of 6 in. from the beginning of the ignition zone, which ordinarily would be the maximum desirable depth of active fuel bed. Nicholls and Eilers, however, took their curves with very deep beds, and the slope of the temperature curves beyond the 6 or 8-in. point could be used to calculate the thermal conductivity of the coke used in their tests.

Fig. 11 shows the rates of ignition and rates of burning obtained in an experimental underfeed fuel bed, as a function of the rate of primary air flow. The curved lines represent the rates of ignition observed with different sizes of fuel, while the straight sloping line represents the rate of burning, which is not affected by the size of the fuel below the intersection with the rate of ignition line. Thus the rate of burning on an underfeed fuel bed is proportional to the rate of primary air flow until it becomes as great as the rate of ignition, when the latter limits the possible rate of burning. It is apparent that commercial stokers are limited to the region to the left of the intersection.

A simplified analysis based on the theory outlined above indicates that the rate of advance of the ignition zone, the difference between the observed rates of ignition and burning, is proportional to the temperature difference between the active combustion zone and the point of ignition and to the difference between the thermal diffusivity of the fuel bed and a factor proportional to the air flow. It is known from previous work(11) that the thermal diffusivity increases rapidly with decrease in size of the fuel. This fact accounts for the observed increase in rate of ignition obtained with a decrease in fuel size for a given primary air rate. The fact that the ignition curves of Fig. 11 show a maximum is accounted for by the fact that the thermal diffusivity appears in a term containing the difference between it and the rate of air flow. As long as this difference is positive the rate of advance of the ignition zone is positive; but when the rate of air flow becomes too great, more heat is abstracted from the coal ahead of the ignition zone by the air than is transferred to it from the hotter parts of the bed, and the rate of advance of the ignition zone becomes negative so that the fire eventually is quenched. The rate of burning in Fig. 11, which is equivalent to what has been called here the rate of advance of the end of the burning zone, is directly proportional to the rate of primary air flow, in complete accord with the result deduced above. Hence the intersection of the curve of the rate of burning with the rate of ignition is a practical limit to the capacity of a grate. It is not, however, to be inferred that the values of this limit obtained by Nicholls and Eilers represent absolute characteristics of the fuels. They depend also on the geometry of the apparatus used, especially as it affects the temperature distribution. It is unfortunate that temperature measurements throughout the bed obtained concomitantly with the measurements of ignition and burning rates were not published, since these might make possible the calculation of the limits of rating in other types of apparatus.

It is evident that it is not possible at present to specify the furnace temperature and volume requirements for a grate fire as completely as could be done for pulverized coal. Evidently the rate of ignition does not depend so directly on the furnace temperature as it did in the powdered-coal furnace because the region of highest temperature, from which heat is conducted back to the ignition zone, is screened from the furnace itself by a conducting layer. Hence the ignition time might be expected to vary less rapidly than inversely as the fourth power of the furnace temperature in a grate fire. The furnace volume must be great enough to insure the combustion of the gas mixture rising from the fuel bed, but as this is a problem in gaseous combustion, rather than in combustion of solid fuel, it will not be considered further.

EFFECT OF COAL CHARACTERISTICS

From the mechanisms postulated above certain conclusions may be drawn concerning the effect of the different characteristics of various coals on the attainable rates of combustion. In pulverized-fuel firing, we have seen that the mechanical features of the environment are considerably more important than the coal characteristics, but certain of the latter have an appreciable effect on the time required for combustion. First to be considered are the quantity and type of volatile matter. We have seen that the expelled volatile matter may considerably increase the rate of heating of the fuel particle if it is of such a nature as to be ignited while it is still close to the particle. In some of the early powdered-coal installations excessive water-cooling of the furnace was introduced, making ignition largely dependent on this effect. Thus some difficulty was frequently experienced in maintaining ignition with low-volatile fuels. Nowadays this effect is less important, especially when the large radiant surface of a molten slag bed is introduced, as in wet-bottom furnaces. The other factor affecting the length of the heating period is the reactivity, since it determines how far external heating of the fuel must proceed before the self-heating represented by *A-B* of Fig. 2 can take place. That these combined effects may be appreciable is shown by Fig. 12, taken from a paper by R. A. Sherman.(12) It will be shown below that great differences in the rates of active combustion of different fuels are not to be expected when they are ground to the same size and fired in the same equipment, and this conclusion is borne out by the fact that the magnitude of the differences between burning time of the different fuels does not change greatly as combustion proceeds. Hence, the greatest part of the observed differences in these burning times, amounting to as much as 30 per cent of their average value, must be due to differences in the length of time required to ignite the fuels.

The only factors that can affect the rate of active combustion of suspended fuel are the density of the fuel and its ash content, but both of these effects are probably small. The density affects the rate of combustion by its influence on the radius of a particle of a given weight, but this effect is largely compensated by the association of the relative velocity between fuel and air with the density. Hence the rate of active burning is expected to vary inversely as a small power of the density. An approximate calculation shows this power to be of the order of ⅙. The effect of the ash is somewhat more difficult to predict. In general, the particle temperature will be well above the melting point of the ash, so that usually the globules of slag will fall or be blown off the particle. This would account for the extremely small size of the dust particles in fly ash. Sometimes the ash may not be fluid enough to drop from the coal particle and occasionally this may result in coating of the fuel particle by slag, which slows or prevents complete combustion.

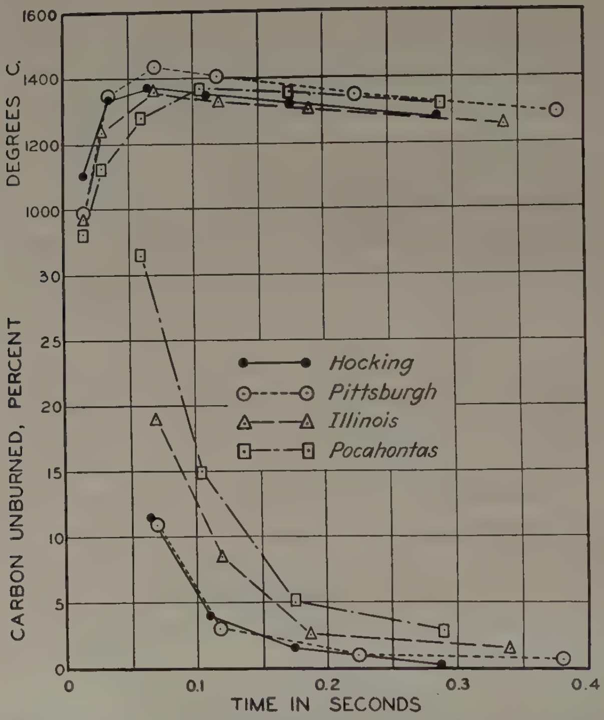

FIG. 12.—RATES OF BURNING OF PULVERIZED COALS (*From R. A. Sherman*[12]). Rate of heat input, 2.7 × 10^6 B.t.u. per hour; 20 per cent excess air; fineness 77 per cent through 200 mesh.

With fuel fired on a grate, the ultimate limit to the rating is fixed by the rate of advance of the ignition zone, which is a function of the thermal diffusivity and of the reactivity of the coke. Thus, in operating parlance, the rating is limited to that at which green coal is not passed to the ashpit. However, a practical limit, in that the fuel bed may become unstable and cause excessive cinder loss, may enter before this point if the coke formed is light and friable. This limit depends on the free burning qualities or the agglutinating value(13) of the coal.

Unless it causes clinkering, the ash can have no effect on the combustion rate of a stoker. Clinker formation effectually prevents the attainment of high combustion ratings, not only because it interferes with the free flow of fuel on the stoker but also because it blocks off air passages. "Combustion catalysts," which are in effect simply additions to the incombustible content of a fuel, probably have little effect on the speed of combustion. It is true that alkali salts may increase the reactivity of cokes, but a considerable amount is required, from 1 to 5 per cent of the weight of the coke, to obtain an appreciable effect, even if it remains in the coke after distillation. On the other hand, some "catalysts" might cause an increase in the ash-fusion temperature, thus decreasing clinkering difficulties. These, however, could not properly be called "combustion catalysts," although they would permit the attainment of higher ratings by decreasing operating difficulties.

SUMMARY

The ultimate limit to the attainable rating in both pulverized-coal firing and grate firing is set by the reactivity of the fuel in its coked form. With pulverized fuel this limit seldom is reached; a practical limit is set by the speed of the active burning process, which is practically independent of the characteristics of the fuel and depends only on the mechanical arrangements for firing. In grate firing the reactivity and the thermal diffusivity enter in the same way, so that both must be high to attain high rating. In addition, the fuel must be strong and dense or a lower limit to the rating will be set by the permissible cinder loss. Thus for high rating with pulverized fuel, the coal must be reactive, or contain a reasonable amount of easily ignitable volatile material. For high ratings on a grate, the coal must yield a reactive coke having high thermal diffusivity in bulk; and it must either be free burning or have high agglutinating power.

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DISCUSSION

(*H. J. Rose presiding*)

S. P. BURKE,* Morgantown, W. Va. (written discussion).—Mr. Mayers is to be commended for giving an able and carefully coordinated résumé of the present state of our knowledge concerning the mechanism of the combustion process of coal, both in the pulverized form and in the lump form. I have discussed, in the references cited by Mr. Mayers and in others, many of the subjects treated in this paper so far as combustion of pulverized fuel is concerned. A detailed discussion therefore would quickly become a repetition of what is already in print. Yet there are certain points in this paper that I wish to amplify.

In modern, properly designed pulverized-fuel furnaces, the volatile matter ignites and burns about as rapidly as it is expelled from the coal particles. It is interesting in this connection to refer to the work of Griffin, Adams and Smith, which indicates that for Kanawha coal the volatile did not ignite when the coal was dropped into air at a temperature of 800° C., but did ignite and burn immediately around the particles when the temperature of the air was 1000° C. The evidence from the meager available analyses on industrial furnaces also indicates that there is little, if any, unburned volatile in the flames of most pulverized coal. The results of Mr. Sherman's work at Battelle Institute on the combustion of pulverized fuel also appear to confirm this statement. Generally speaking, therefore, unless the temperature of the furnace is lower than that usually encountered in industrial practice, the volatile of the coal will burn around the surface of the particle as it is expelled therefrom. It is obvious, of course, that if the admixture of air with the coal is deficient, the volatile on being expelled from the coal will not be able to burn until it has contacted sufficient air for that purpose. In this event, combustion of the volatile may be substantially removed from the particles themselves. In modern installations, this condition does not seem to be encountered.

However, Mr. Mayers has chosen to consider the case where, for various reasons, the volatile does not burn immediately on evolution. Under these conditions, I believe that Fig. 1 should be altered. At about 350°, the curve should break and should reach 600° at about 160 milliseconds or more. This conclusion is based on the fact that the rate of absorption of heat by the particle would be substantially constant from 100° to 600° C. However, the amount of heat required to raise the temperature through the second 250°, that is, from 350° to 600° C., is about twice as great as that required to raise it to the first 250°, that is, from 100° to 350° C.[14]

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* Chairman of Graduate Council and Director of Industrial Science Division, West Virginia University.

[14] S. P. Burke and V. F. Parry: The Heat of Distillation of Coal. *Ind. & Eng. Chem.* (1927) **19,** 15.

If, on the other hand, the volatile burns immediately around the coal, the rate of rise during the second 250° will be at least as rapid and probably more rapid than that shown in Fig. 1. It is also interesting to observe that when the volatile does burn under these conditions, the extreme temperature generated by the flame and the resultant high viscosity of the gases would probably cause the combustion to take place in accord with the laws of "diffusion flames[15]." In any event, it is clear from the photographs of Griffin, Adams and Smith that at the termination of the combustion of the volatile the residual particle of coal is relatively cool for a period of perhaps 10 to 40 milliseconds. It is during this period that the reactivity of the coke cenosphere is significant, for, based upon this reactivity, this period of heating up will be short or long. It is my opinion, based upon no data, however, that the reactivity of coke or char formed under these conditions from practically all bituminous coal is such that when the solid is bombarded by oxygen molecules at the temperatures normally used in industrial furnaces, this period of slow combustion or heating up period will rarely exceed 50 milliseconds for particles of this size. At the conclusion of this period, the diffusion combustion of the carbon has set in. On the basis of this picture, and the data available to the writer, a modification of Fig. 2 is suggested. A break should occur in the temperature curve at about 350°, the curve sloping off and reaching 600° at about 160 milliseconds. Within a space of 50 milliseconds or less, the temperature of the particle would then rise to something appreciably in excess of 2000° C. Combustion of the particle the size Mr. Mayers has taken (50μ = 300 mesh approximately) would then be completed within about 50 milliseconds and the temperature graph during this latter phase would curve *upwards*. What actual temperature would be reached is impossible to say. Combustion would thus be completed in about 0.26 sec.; at the end of that time, there would remain the particle of ash initially present in the coal, which of course would quickly cool. If, however, an unlimited amount of air were not present, but instead combustion were being conducted with approximately 130 per cent of the theoretical amount of air, the picture given above would be unaltered except that the time for the active burning of the carbon residue by diffusion would be lengthened to approximately double that indicated above; that is, to approximately 100 milliseconds. In practice, however, where the volatile burns in proximity to the particle, the entire time from admission to complete combustion would be about 200 milliseconds, using 130 per cent air.

I base my contention of the temperature of the particle on the fact that both theory and experiment indicate that carbon particles burning under these conditions increase in temperature as the particle grows smaller. This rate of increase becomes astonishingly rapid when the particle becomes 100 mesh or smaller in size. With carbon particles about ½ mm. in diameter burning in air at about 1000° C., I have measured surface temperatures in excess of 1650° C. The temperature attained by particles of the size Mr. Mayers is discussing must be substantially greater than this. It is to be noted that Sherman (ref. 12) records temperatures of 1400° C. in pulverized-coal flames. Considering the circumstances and the method used, it is certain that the actual temperature of the surface of the fuel particles was several hundred degrees above that recorded by Sherman's thermocouple. My statement regarding the time during which the diffusion combustion of the carbon residue takes place is based upon the calculations given in my publications to which Mr. Mayers refers and also to the fact that the findings of Griffin, Adams and Smith and of Sherman in the various publications referred to seem to be in substantial accord with these conclusions.

During the diffusion combustion of the carbon residue, practically no oxygen impinges on the fuel surface, but the combustion occurs substantially by the reaction of carbon dioxide to form carbon monoxide and the subsequent combustion of the carbon monoxide in a sphere of flame at a very slight distance from the surface of the carbon.

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[15] S. P. Burke and T. E. W. Schumann: Diffusion Flames. *Ind. & Eng. Chem.* (1928) **20,** 998.

In the first of my papers referred to by the author, the relation between time of combustion and excess air was computed. On the basis of the hypothesis advanced in that paper and of the computations made, I am in substantial agreement with the author except that my deductions lead me to the conclusion that the ratio of the time of combustion for various percentages of excess air is also a function of the initial size of the carbon particle being burned.

I cannot fully agree with Mr. Mayers in his discussion of the advantages of turbulence in the pulverized-fuel furnace. Turbulence, of course, is highly important and necessary, to bring the required 20,000 volumes (or thereabouts) of air in proximity to the coke particle and its encasing gaseous film. Since, however, the density of the fuel particle together with its attendant envelope of gases is so nearly the same as that of the general furnace atmosphere, it would appear that little can be done toward reducing the thickness of the films around the particle by turbulence. This factor is more completely discussed in the first of my papers referred to by the author.

The author attributes the difference in "burning times" of various coals, as shown by the data of Sherman (Fig. 12) to differences in reactivity. I believe that this difference is due primarily to the densities of the carbon residues after the volatile has escaped. It has been shown (ref. 3) that for a particle of given radius the time of active burning is proportional to the density (not, in my opinion, to the $\frac{1}{6}$ power of the density). Pittsburgh and Hocking coals lose most weight as volatile and in addition the coke residues swell and form cenospheres. Illinois and Pocahontas coals swell much less. Moreover, the particle of the Pocahontas coal loses much less weight as volatile. Thus the combustion times of these coals are placed in the observed order without recourse to a hypothetical difference in reactivity.

Concentration of Banded Ingredients of Illinois Coals by Screen Sizing and Washing

By L. C. McCabe,* Member A.I.M.E.

(New York Meeting, February, 1936)

This paper is a progress report on a study of the distribution of the banded ingredients (Figs. 1 and 2) in Illinois screenings and the method of determining their distribution. Proximate analyses and screen tests[1] have been reported and a further report on the washability characteristics of the screenings is in preparation. The microscopic study of the screenings is not yet far enough advanced to permit a conclusive report on the effect of mining, screening and washing on the distribution of the banded ingredients in the coals of Illinois. However, considerable fundamental data have been collected as a background for the screenings investigation and for a proposed study of fuels composed of known quantities of vitrain, clarain and fusain, the three common banded ingredients of Illinois coals. These fundamental data and the preliminary information now available from the microscopic study of screenings are broadly indicative of the reaction of the coals to the physical processes affecting them in mining and preparation.

Proximate analyses of samples of vitrain, clarain, and fusain from several Illinois mines have been published[2], but those given in Table 1 serve to show the differences that are apparent in proximate analyses. The vitrain samples are consistently low in ash, and the unit-coal value is uniformly lower than in clarain. The average fuel ratio (F.C./V.M.) of the vitrain samples in the table is 1.57 whereas the clarain average is 1.42. The fusain analyses vary considerably in ash and moisture. In the two samples from this mine the moisture values range from 8.99 to 22.80 and the ash from 4.86 to 16.91 per cent. Fusain has higher fixed carbon than either vitrain or clarain but the percentage is erratic from sample to sample.

The relative proportions of the banded ingredients in the Herrin (No. 6) coal in Illinois is reasonably well known but detailed information

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Manuscript received at the office of the Institute Dec. 2, 1935. Published by permission of the Chief, Illinois State Geological Survey.

* Assistant Geologist, Coal Division, Illinois State Geological Survey, Urbana, Ill.

[1] L. C. McCabe, D. R. Mitchell and G. H. Cady: Proximate Analyses and Screen Tests of Illinois Coals. Illinois State Geol. Survey *R.I.* No. 38 (1935).

[2] L. C. McCabe, D. R. Mitchell and G. H. Cady: Banded Ingredients of Illinois Coals and Their Heating Value as Related to Washability Characteristics. Illinois State Geol. Survey *R.I.* No. 34 (1934).

for the other workable coal beds is limited to a very few mines. Clarain, however, is the predominant constituent in all coal beds (Table 2).

TABLE 1.—*Analyses of Banded Ingredients*

County	Analysis No.	Ingredient	Condition[a]	Moisture	Ash	Volatile Matter	Fixed Carbon	Sulfur				B.t.u.
								Sulfate	Pyritic	Organic	Total	
Franklin (No. 6 coal).	C-510	Vitrain	1	8.87	0.87	35.60	54.66	0.00	0.18	0.49	0.67	13,010
			2		0.95	39.07	59.98	0.00	0.20	0.53	0.73	14,277
			3			39.44	60.56	0.00	0.20	0.54	0.74	14,415
			4									14,447
	C-509	Vitrain	1	9.63	0.88	33.84	55.65	0.00	0.13	0.56	0.69	12,866
			2		0.97	37.45	61.58	0.00	0.15	0.61	0.76	14,237
			3			37.81	62.19	0.00	0.15	0.62	0.77	14,376
			4									14,411
	C-503	Clarain	1	8.6	6.7	35.9	48.8	0.00	0.21	0.48	0.69	12,269
			2		7.3	39.2	53.5	0.00	0.23	0.52	0.75	13,426
			3			42.3	57.7	0.00	0.25	0.56	0.81	14,486
			4									14,600
	C-504	Clarain	1	8.9	5.6	34.5	51.0	0.00	0.33	0.50	0.83	12,372
			2		6.1	37.9	56.0	0.00	0.36	0.55	0.91	13,582
			3			40.4	59.6	0.00	0.38	0.59	0.97	14,471
			4									14,569
	C-505	Fusain	1	8.99	16.91	22.35	51.75	0.01	0.24	0.15	0.40	9,335
			2		18.58	24.56	56.86	0.01	0.26	0.17	0.44	10,257
			3			30.16	69.84	0.01	0.32	0.21	0.54	12,597
			4									12,844
	C-506	Fusain	1	22.80	4.86	6.78	65.56	0.01	0.21	0.13	0.35	10,737
			2		6.29	8.78	84.93	0.01	0.27	0.17	0.45	13,908
			3			9.37	90.63	0.01	0.29	0.18	0.48	14,842
			4									14,937

[a] 1. Moist bands as collected in the mine.
2. Moisture-free.
3. Moisture-and-ash-free.
4. Unit coal B.t.u. (dry, mineral-matter-free).

The fusain content of Herrin (No. 6) coal is highest on the western margin of the field and lowest toward the center and deeper parts of the basin. The clarain content is lowered by the presence of extraneous impurities and in two instances at least by the presence of durain. If banded impurities are excluded from the calculation, clarain and vitrain become essentially complementary. Vitrain shows such a marked increase in quantity from the Belleville region in St. Clair County to Franklin County that it is possible to represent this change in the banded character of the coal by lines of equal vitrain content (Fig. 3).

Mine I, one of the ten mines where screenings were sampled, was in the high-vitrain area of Herrin (No. 6) coal in Williamson County. A gross sample of 1345 lb. was taken from the loading chute in 12 to 20-lb. increments at 5-min. intervals throughout a day's operation. The

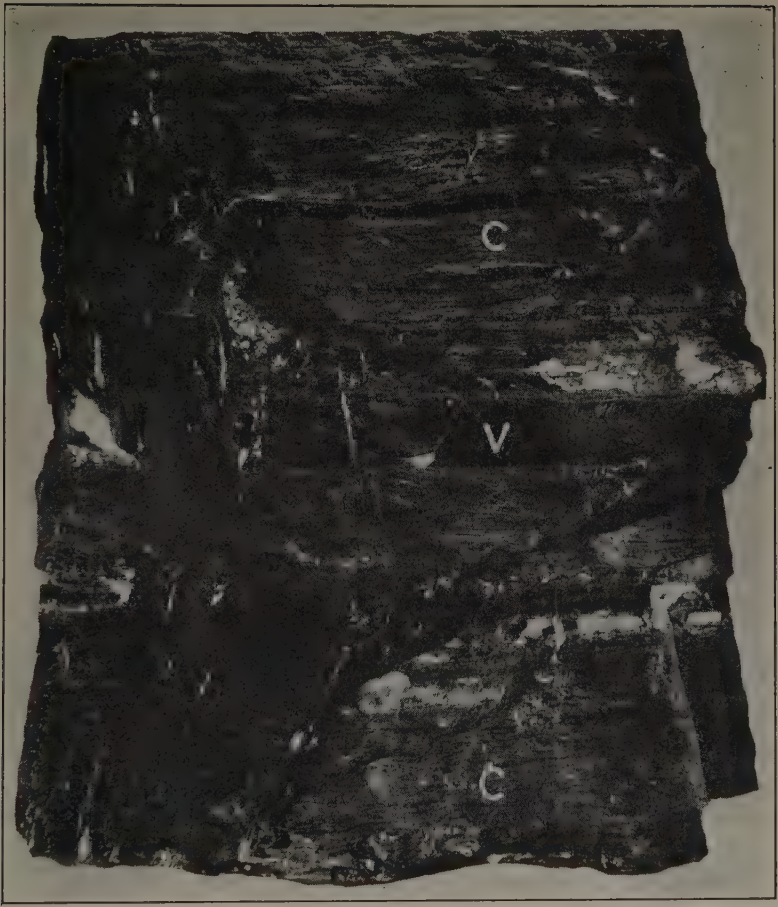

Fig. 1.—Typical block of No. 6 coal, showing two varieties of banded ingredients, vitrain (*V*) and clarain (*C*). Natural size.

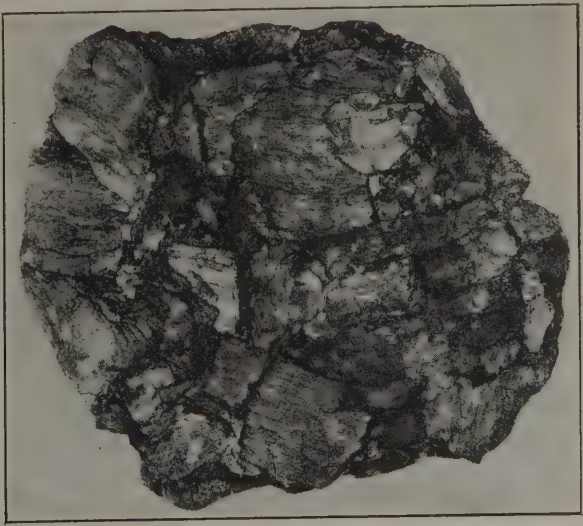

Fig. 2.—Block of coal composed partly of mineralized fusain.

TABLE 2.—*Percentages of Banded Ingredients in Columnar Sections of Coal Beds* 2, 5 *and* 6

Each value represents an average of two determinations.

Coal Bed No.	County	Banded Coal Ingredients				Banded Impurities		
		Vitrain	Clarain	Durain	Fusain	Pyrite	Bone	Clay
2	Fulton	17.75	78.52		0.56	2.05		1.12
5	Saline	26.60	69.30		1.80	2.30		
6	St. Clair	13.16	76.51		3.38	5.12		1.83
	St. Clair	7.83	83.09		2.62	4.76		1.70
	St. Clair	9.95	77.58		7.43	1.82	1.33	1.89
	Montgomery	19.5	62.1	14.9	0.8	1.5		1.2
	Washington	13.40	68.70	15.0	2.9			[a]
	Randolph	14.76	75.88		4.77			4.59
	Perry	19.00	69.90		4.50		2.6	4.00
	Franklin	18.81	77.68		1.50	0.42	0.61	0.98
	Franklin	22.70	72.40		2.3	0.10	1.60	0.9
	Williamson	20.10	76.55		1.35		2.0	

[a] Eight-inch clay band removed in mining.

sample was riffled into quarters in the laboratory; one quarter was kept as a reserve, another was crushed to furnish an over-all sample for proximate analysis, a third quarter was screened into five sizes and crushed to furnish samples for proximate analysis, and the fourth was similarly screen sized and each size further separated by float-and-sink methods. Sizes made were as follows: 1¼ to ¾ in.; ¾ to ⅜ in.; ⅜ in. to 10 mesh; 10 to 48 mesh; and minus 48 mesh. Round-hole screens were used in sizing at ⅜ in. and above and Tyler standard sieves for sizing below ⅜ in. Float-and-sink separations on each size were at 1.30, 1.35, 1.40, 1.50 and 1.70 sp. gr. Zinc chloride solutions were used in separating 1¼ to ¾-in. and ¾ to ⅜-in. sizes; benzene, carbon tetrachloride and bromoform mixtures were used in separating the three smaller sizes. The smaller sizes were separated in small quantities to reduce the possibility of mechanical entanglement and fractions were not refloated. The float-and-sink fractions from the three largest sizes were fed to rolls set to crush to 3⁄16 (0.1875) in., as it was desired to have all particles pass the Tyler 3-mesh screen with an opening of 0.263 in. Two samples were riffled from each float-and-sink fraction to furnish material for microscopic and proximate analyses.

The samples for microscopic assay of the banded ingredients were prepared by screening each float-and-sink fraction through the sieves in the Tyler scale where openings increase in the ratio of the square root of 2. All screens in the series between 200 and 3 mesh were used. An ore-dressing microscope with a fixed counting tray and movable binocular body was used for assay count. Two hundred to five hundred grains was riffled from each sieve for the distribution determination. In establishing a weight per cent for each ingredient it was assumed that all particles on a screen were of the same size and gravity.

The microscopic assay of the distribution of the banded ingredients floating on a solution of 1.30 sp. gr. (Table 3) shows an increase in the percentage of vitrain with a decrease in screen size. The vitrain percentage of 42.3 in the 1¼ to ¾-in. size is essentially doubled in the minus 48-mesh coal. This concentration of vitrain in the finer sizes in coal of the same specific gravity is undoubtedly due to the high friability of vitrain as compared to clarain. In this connection it is noteworthy that the clarain percentage of 55.6 in the 1¼ to ¾-in. coal is reduced to 17.9 per cent in the minus 48-mesh coal.

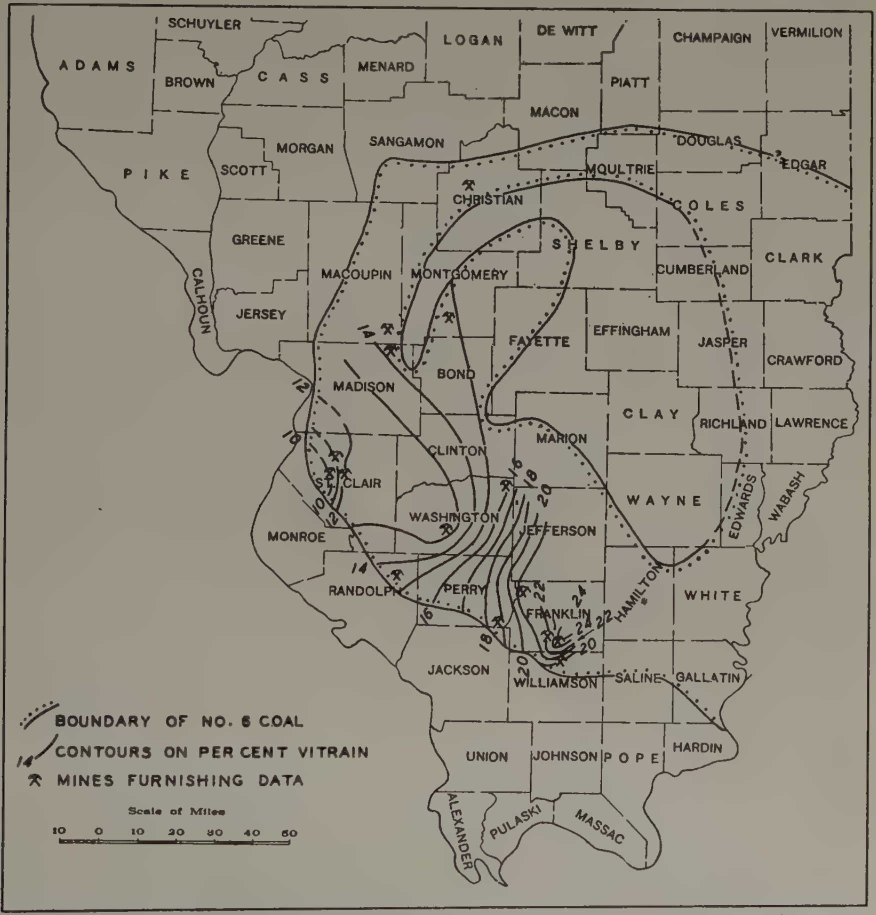

FIG. 3.—GENERALIZED VITRAIN DISTRIBUTION IN NO. 6 COAL, SOUTHERN ILLINOIS.

Of the minus 1¼-in. screenings from this mine, 58.9 per cent float at 1.30 sp. gr. The coal floating at 1.30 sp. gr. is 58.2 per cent vitrain, 40.0 per cent clarain, 1.1 per cent fusain, and 0.7 per cent middling refuse. Table 2 shows the percentages of the banded ingredients in the bed for the Williamson County mine as follows: 20.10 per cent vitrain, 76.55 per cent clarain, 1.35 per cent fusain and 2.0 bone (middling refuse). The two percentages are not strictly comparable because the distribution of banded ingredients in the bed given in Table 2 was determined on the polished surfaces of two columns from the mine, and measurement of vitrain was limited to visible bands 0.5 mm. thick, while in the screening samples measurements were made with a microscope at magnifications of three to four times on the larger grains and at 75 times on the dust sizes. This difference in method probably does not invalidate the comparison, however, as by far the greater weight per cent of all samples remained on the coarser sieves. The average vitrain content of the coal bed is only about 20 per cent. This vitrain content is so highly concentrated in the fraction floating at 1.30 sp. gr. that there must be a diminution of vitrain and an increase of clarain in the nut and large sizes.

TABLE 3.—*Percentage of Coal Floating at* 1.30 *Specific Gravity and Distribution of the Banded Ingredients* (*Mine I*)

Size	Wt. Per Cent of Size (Dry)	Wt. Per Cent of 1¼ to 0 Feed (Dry)	Ash Per Cent (Dry)	Vitrain Per Cent	Clarain Per Cent	Fusain Per Cent	Middlings Refuse Per Cent
1¼ to ¾ in.....	54.13	15.54	4.4	42.3	55.6	1.6	0.5
¾ to ⅜ in......	53.56	12.55	3.7	60.4	38.3	1.1	0.2
⅜ in. to 10-mesh	77.70	20.87	4.6	63.8	35.1	0.6	0.5
10 to 48 mesh....	69.33	8.86	3.5	67.8	28.8	1.6	1.8
Minus 48 mesh..	17.87	1.08	1.7	79.3	17.9	0.7	2.1
Total or average........		58.90		58.2	40.0	1.1	0.7

In the microscopic study of the fractions floating at 1.30 sp. gr. a grain of mineral matter or a piece of coal of extremely dull appearance was occasionally observed. All such particles were isolated and placed in a solution of 1.50 sp. gr. If the particle sank it was classed as *middling refuse*. In the larger sizes these heavy particles were probably present in the cleanest coal as banded impurities or mineral facings and were broken free when the coal was crushed for microscopic study. In the smaller sizes such particles were probably free in occurrence but were entrapped in the float coal. This is borne out by the observation that as the size of the particles decreases there are fewer pieces of coal of mixed classification. Single particles in which two or more of the banded ingredients or a banded ingredient and refuse are represented are rarely observed in the sizes below 10 mesh. In spite of the relatively high percentage of middling refuse in the minus 48-mesh coal of the 1.30 sp. gr. fraction, the ash percentage is very low. This may be explained by the concentration of vitrain, which is inherently low in ash (Table 1).

Considering each size individually, it is apparent from column one of Table 3 that the greatest weight per cent of coal floating at 1.30 sp. gr. is in the ⅜-in. to 10-mesh size. In all mines studied in the screenings investigation, the percentage of float at 1.30 sp. gr. in the ⅜-in. to 10-mesh size is consistently higher than in any other size (Table 4, Fig. 4). The reasons for the high percentage of float in this size are not yet clearly determined. Before the microscopic study of the fractions in the 1.30 sp. gr. range was completed, the possibility that the highest vitrain concentration would be found in the ⅜-in. to 10-mesh size suggested itself, but, as Table 3 shows, this appeared not to be the correct explanation, as the vitrain was found to be more highly concentrated in the two

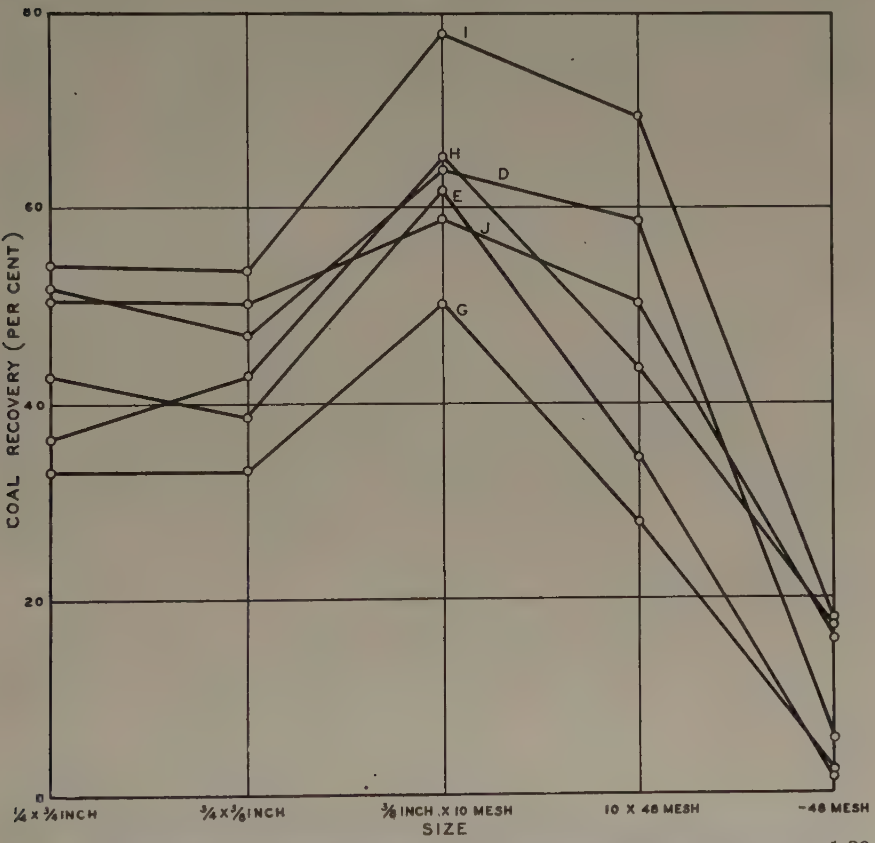

Fig. 4.—Diagrammatic representation of coal recovery in various sizes at 1.30 specific gravity. Letters designate mines.

smaller sizes. That it was not consistently due to low ash content is evident. Table 4 shows the ash to be at a maximum in the ⅜-in. to 10-mesh in the samples from mines H and I, and higher than in the next larger size (¾ to ⅜-in.) for the samples from mines E, G and J. Many pieces of the ⅜-in. to 10-mesh, 1.30 sp. gr. coal from mine I, before crushing for microscopic analysis, were composed of vitrain with attached clarain. It is possible that this clarain, while it does not sink at 1.50 sp. gr. after the sample is crushed, and is therefore not classed as middling refuse, is carried up into the 1.30 sp. gr. float fraction by the much lighter vitrain. This suggestion, however, has not been verified.

TABLE 4.—*Percentages of Float and Ash at* 1.30 *Specific Gravity in Different Mines and Coal Beds*

Mine	D		E		G		H		I		J	
County	Vermilion		Sangamon		St. Clair		Marion		Williamson		Saline	
Coal Bed No.	5		5		6		6		6		5	
Size	Float	Ash	Float	Ash	Float	Ash	Float	Ash	Float	Ash	Float	Ash
1¼ to ¾ in.	51.74	5.8	42.78	7.0	33.03	5.6	36.35	5.1	54.13	4.4	50.54	5.3
¾ to ⅜ in.	46.92	5.2	38.72	6.2	33.10	4.5	42.78	5.1	53.56	3.7	50.13	4.8
⅜ in. to 10 mesh	63.86	4.4	61.90	6.5	50.01	5.2	65.29	5.5	77.70	4.6	58.96	4.5
10 to 48 mesh	58.64	3.7	34.42	3.5	27.89	3.2	43.66	3.6	69.33	3.5	50.12	3.2
Minus 48 mesh	5.68	2.0	1.53	2.2	2.38	1.4	17.12	3.3	17.87	1.7	15.88	2.0

TABLE 5.—*Occurrence of Banded Ingredients and Specific Gravity Distribution of Coal*

1¼ × ¾-inch Size, Mine I

	Specific Gravity					
	1.30, Float	1.30–135	1.35–1.40	1.40–1.50	1.50–1.70	1.70, Sink[a]
Weight per cent 1¼ to ¾-in. size	54.13	32.22	5.72	3.99	1.30	2.64
Vitrain	42.3	35.9	31.2	24.9	20.7	
Clarain	55.6	58.8	55.7	43.2	30.5	
Fusain	1.6	1.8	1.3	3.6	6.0	
Middling refuse	0.5	3.5	11.8	28.3	42.8	

[a] Refuse not studied microscopically.

The individual pieces of 1¼ to ¾-in. coal from mine I were composed entirely of clarain or of clarain interbanded with small vitrain layers or with fragments of fusain. Table 5 shows that the coal with the greatest vitrain percentage floats at 1.30 sp. gr. and that the lowest percentage of vitrain is in the range of 1.50 to 1.70 sp. gr. The clarain percentages are high in the first three fractions but decline as the middling refuse increases.

The fact that the Herrin (No. 6) coal in Williamson County is low in fusain (Table 2) is no doubt reflected in the sizes and float-and-sink fractions reported upon. Table 3 shows no concentration of fusain due to sizing in the 1.30 sp. gr. float. In the minus 200-mesh coal of this gravity, which was screened from the minus 48-mesh sample, the fusain content was less than one per cent. However, in the 1¼ to ¾-in. material the fusain tends to concentrate in the fractions of highest specific gravity (Table 5). Microscopic examination shows this fusain to be highly mineralized.

The literature on the banded ingredients is rapidly growing. Preparation for or actual small-scale use of the separated ingredients in the manufacture of domestic and electrode coke, in hydrogenation, and for diesel fuel has been reported by European investigators. The greatest obstacle to closer control of the character of the coal, in so far as it may be brought about by regulating the distribution of the banded ingredients, is in the relatively low price received for the coal and the high cost that would be involved in special methods of preparation. The systematic study of coal employing the methods used in the best metal-mining practice may throw much light on the constitution and behavior of the fuel. The data presented here are not conclusive, as much proving with the microscope remains to be done before the response of the coal to physical handling can be predicted.

Investigation of Procedure for Determination of Coal Grindability by the Ball-mill Method

By C. G. Black,* Junior Member A.I.M.E.

(New York Meeting, February, 1936)

The purpose of this paper is to present data obtained from an investigation conducted on the grindability of coal by the American Society for Testing Materials Tentative Standard ball-mill method. This investigation has been conducted over a period of a year, during which time several hundred tests were run on a wide selection of different coals coming from Pennsylvania, Ohio, West Virginia and Kentucky.

The data herewith presented tend to show that the tentative standard ball-mill method might give results that are misleading and that a volume measure might be more indicative of the true grindability of materials of varying specific gravity. A means whereby the time element might be reduced has been studied and other phases of this work such as preparation of samples, mechanical screening, reporting results and duplicating tests have been investigated. It is hoped that all criticism offered in this paper will be taken as constructive, with the purpose in mind of perfecting the method now tentative to show the true conditions and of making optional the manner in which certain parts of the test might be conducted to meet diversified laboratory conditions.

Weight and Volume

The results of grindability on extraneous material from the coal seam obtained with the present tentative standard ball mill method indicate easier grinding of these materials as compared to most coals. Varying amounts of these impurities in the coal therefore may cause a raising of the grindability index and may lead to the erroneous conclusion that coals containing large amounts of these foreign materials are easier to grind. For a true comparison of grindability, it would seem necessary to grind to the same particle weight. If material were sized in a commercial mill by means of an air current set at a velocity to remove 1.60 sp. gr. float coal ground to 80 per cent minus 200 mesh, any 1.60 sink particles would be left behind, and in order to remove this 1.60 sink material by the same current it would be necessary to grind finer and produce a

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particle weight equal to the particle weight of the 1.60 float coal. A test of this nature is impossible but it can readily be seen that a coal with a smaller amount of heavy-gravity foreign material should be more favorable in grindability than the same type of coal containing large amounts of these foreign substances.

Directly dependent on the particle weight would be new surface area produced. A measure of this would show that for equal weights of two materials of different specific gravity reduced to the same degree of fineness the one of lower specific gravity would have a surface area larger than the one of higher specific gravity. We believe that the measure of surface area of this fine material should be avoided because it involves questionable surface factors and fine screening, which is inaccurate and expensive. However, the new surface area produced is dependent on original surface area. Equal weights of two materials of different specific gravity of same particle size will contain different number of particles and therefore different surface areas. Equal surface areas or the same number of particles of the same particle size will produce the same volume. The nearest practical measure of original surface area would be a volume measure and when equal volumes of particles of the same size are reduced to the same degree of fineness, equivalent surface areas should be produced.

Thus it can be seen that in order to get the true grindability of materials of varying specific gravities, it is necessary to grind either to the same particle weight or to the same surface area. The former is impossible; the latter can be performed by taking the same surface areas as determined by a volume measure and grinding to a certain degree of fineness, which should produce the same final surface areas. It would therefore seem advisable to conduct the present method by taking constant original volumes and grinding them to 80 per cent minus 200 mesh by weight, and the following is submitted in support of this claim.

Purpose and Summary of Results

Purpose.—To determine the grindability of specific gravity fractions of coal and mixtures of 1.60 sp. gr. float and 1.60 sink materials by taking a given initial weight and volume of materials and grinding them to 80 per cent minus 200 mesh by weight, using the tentative standard ball-mill method.

Summary of Results.—Grindability as determined from constant weight shows a decrease in the number of revolutions required to grind 80 per cent minus 200 mesh in going from the lower specific gravity fractions to the higher specific gravity fractions. When a constant initial volume is used, it shows an increase in the number of revolutions in going from the lower to the higher specific gravity fractions.

When mixtures of 1.60 float and 1.60 sink material are run for grindability by the constant-weight method (the tentative standard ball-mill method), there is a decrease in the number of revolutions required as the amount of 1.60 sink material increases. When these mixtures are run by the constant-volume method, there is an increase in the number of revolutions required as the amount of 1.60 sink material increases.

Method for Determining Volume

The volume was determined by taking a 500-gram sample of 10 by 200-mesh material, thoroughly rolling it on a rubberized cloth, approximately 2 ft. square, and "pouring" it from the cloth into a 1000-c.c. graduate calibrated to 10 c.c. The material was leveled in the graduate by means of a ½-in. brush, care being taken not to jar or compress the material, and the graduate was read to the nearest 10 c.c. Check results are very easily obtained, both on the same sample and samples of the same coal prepared at different times.

Volume was determined by loose measure as above and also by tapping to a constant volume, and the same loose volume of material in every instance tapped to the same constant volume. A volume of 750 c.c. of material, regardless of whether it was 1.30 float material or 2.60 sink material "tapped" to approximately 660 c.c. volume. Since it was much easier and quicker to measure a loose volume than a "tapped" volume, a loose measure was used in the grindability work. Table 1 shows the close agreement of the specific gravity fraction of coal *X* on a loose and tapped volume, assigning an index number to 100 to the 1.30 float figures.

TABLE 1.—*Volume of* 500 *Grams* (10 *to* 200 *Mesh*) *Coal X*[a] *Specific Gravity Fractions*

Specific Gravity	Volume 500 Grams Loose		Volume 500 Grams Tapped	
	C.c.	Index No.	C.c.	Index No.
1.30 float	750	100	660	100
1.30–1.40	730	97	640	97
1.40–1.60	650	87	570	86
1.60–2.00	560	75	490	74
2.00–2.60	400	53	355	54
2.60 sink	340	45	300	45
Head	710	95	620	94

Surprisingly close results were obtained for the weight of a 750 c.c. volume of the specific gravity fractions when the weight of a volume of less than 750 c.c. was known, by taking a direct ratio as shown in Table 2.

TABLE 2.—*Calculated and Actual Weights of* 750 *C.c. Volume* (10 *to* 200 *Mesh) Coal X*[a] *Specific Gravity Fractions*

Specific Gravity	Known Volume, C.c.	Weight Known Volume, Grams	Calculated Weight, Grams	Actual Weight, Grams
1.30 float	750	500	500	500
1.30–1.40	730	500	514	513
1.40–1.60	650	500	577	576
1.60–2.00	560	500	670	669
2.00–2.60	400	500	938	926
2.60 sink	340	500	1103	1102
Head	710	500	528	530

[a] ⅜ to 1⅛-in. size.

In taking mixtures of 1.60 float and 1.60 sink of coal *Y*, when the volumes of 500 grams of the 1.60 float and 1.60 sink were known, the calculated volumes of 500 grams of the mixtures of these two specific gravities checked very closely with the actual volumes measured as shown in Table 3.

TABLE 3.—*Calculated and Actual Volumes of* 500 *Grams of Mixtures of* 1.60 *Float and* 1.60 *Sink* (10 *to* 200 *Mesh*) *Coal Y*[a]

Mixture, Per Cent Wt.		Calculated Volume, C.c.	Actual Volume, C.c.
1.60 Float	1.60 Sink		
100	0		740
95	5	727	730
90	10	713	720
85	15	700	700
80	20	686	690
75	25	672	670
50	50	605	605
25	75	537	530
0	100		470

[a] ⅜ to 1⅛-in. size.

From Table 3, 500 grams of 1.60 float has a volume of 740 c.c. and 500 grams of 1.60 sink has a volume of 470 c.c. Therefore, theoretically a 50-50 mixture would have a volume of $0.5 \times 740 + 0.5 \times 470$ or 605 c.c. and by an actual measure was found to be 605 c.c. In the same manner as shown on coal *X* in Table 2, the actual weights of 750 c.c. volumes of mixtures of 1.60 float and sink of coal *Y* check very closely the calculated weights from the weight of a known volume, shown in Table 4.

In determining the volume of the specific gravity fractions of coal *X*, 500 grams of 1.30 float occupied 750 c.c. and this volume was arbitrarily taken as a standard. This matter is not so important, as any volume can be taken as a standard and the results changed to another volume by direct ratio.

TABLE 4.—*Calculated and Actual Weights of* 750 *C.c. Volumes* (10 *to* 200 *Mesh) Coal Y Mixtures* 1.60 *Float and* 1.60 *Sink Specific Gravity Fractions*

Mixture, Per Cent Wt.		Known Volume, C.c.	Weight Known Volume, Grams	Calculated Weight, Grams	Actual Weight, Grams
1.60 Float	1.60 Sink				
100	0	740	500	507	507
95	5	730	500	514	514
90	10	720	500	521	520
85	15	700	500	536	538
80	20	690	500	543	544
75	25	670	500	560	560
50	50	605	500	620	620
25	75	530	500	708	706
0	100	470	500	798	799

If a volume measure should be adopted, it is recommended that a standard measuring container be used by all laboratories. This container should be made of glass, cylindrical in shape, with a height equal to the diameter. For a 750 c.c. volume, this container would be 9.85 cm. in height and in diameter. The volume-weight measure can then be determined after the A.S.T.M. method for determining the weight of a cubic foot of coke; i.e., by filling the container until it overflows on all sides and leveling it by passing a straight edge across the top. The tare and gross weights can be determined to the nearest gram, and the net weight of the 750 c.c. of the material taken as the difference.

Grindability Test Method

All samples were stage-crushed to minus 10 mesh on a McCool pulverizer, which is a disk pulverizer. The tentative standard ball-mill method was followed as to cycle operation except that the minus 200-mesh material was removed by the A.S.T.M. standard method of mechanical screening, which is compared to hand screening under a separate heading. Approximately 10 per cent of the material by weight was removed per cycle and the results obtained by the full eight-cycle test. Several checks were made to duplicate test work and permissible checks were obtained in every case.

Tests were run by constant weight (500 grams) and constant volume (750 c.c.) on specific gravity fractions of coal *X*, which was 3/8 to 1 1/8 in. in size, separated at 1.30, 1.40, 1.60, 2.00 and 2.60 specific gravities. Zinc chloride solution was used for the first four gravities and a mixture of kerosene and acetylene tetrabromide was used for the 2.60 gravity. Mixtures of 1.60 float and sink of coal *Y*, which was also ⅜ to 1⅛ in. in size, were run by both methods at percentages of 0, 5, 10, 15, 20, 25, 50, 75 and 100 of 1.60 sink. Table 5 shows the grindability results expressed as revolutions on the specific gravity fractions of coal *X* determined by the constant-weight and constant-volume methods. It also shows the cumulative revolutions of the gravity fractions. This table indicates that the present tentative ball-mill method shows all three of the heavy gravity fractions easier to grind than the three lighter fractions, while the volume method shows the heavier gravity fractions harder to grind than the lighter coal fractions. A method that shows the 2.00 to 2.60 and the 2.60 sink fractions, which will run around 30 per cent sulfur, considerably easier to grind than coal does not, in our estimation, give the true picture of grindability.

TABLE 5.—*Revolutions Required to Grind Specific Gravity Fractions of Coal X to* 80 *Per Cent Minus* 200 *Mesh by Constant Weight and Volume Methods*

Specific Gravity	Per Cent Weight	Cumulative Per Cent Weight	Constant Weight 500 Grams		Constant Volume 750 C.c.	
			Number Revolutions	Cumulative Number Revolutions	Number Revolutions	Cumulative Number Revolutions
1.30 float	20.4	20.4	1150[a]	1150	1150[a]	1150
1.30–1.40	67.2	87.6	1121	1127	1158	1156
1.40–1.60	3.5	91.1	1187[a]	1132	1400	1164
1.60–2.00	3.0	94.1	1109	1128	1445	1174
2.00–2.60	5.0	99.1	861[a]	1116	1603[a]	1196
2.60 sink	0.9	100.0	700	1112	1505	1199
Head sample			1135		1215	

[a] Checked.

The cumulative number of revolutions is plotted against the specific gravity from data of Table 5 in Fig. 1. This shows that by the weight method the number of revolutions required to grind 80 per cent minus 200 mesh decreases as the specific gravity increases, but by the volume method the number of revolutions increases as the specific gravity increases.

Table 6 shows that when the number of revolutions is known for a volume of less than 750 c.c., the calculated results for a 750-c.c. volume check closely with the results obtained by test. It also shows that for 400 c.c. of 2.00 to 2.60 specific gravity material 861 revolutions are required to grind 80 per cent by weight to minus 200 mesh. Therefore, theoretically, to grind 750 c.c. to the same degree of fineness would require $^{750}/_{400} \times 861$ or 1614 revolutions and by actual test required 1603 revolutions. The calculated number of revolutions on all the gravity fractions check within 3.0 per cent of the actual values on all of the gravity fractions. Table 7 shows how closely the cumulative calculated values from Table 6 check the cumulative actual values from Table 5.

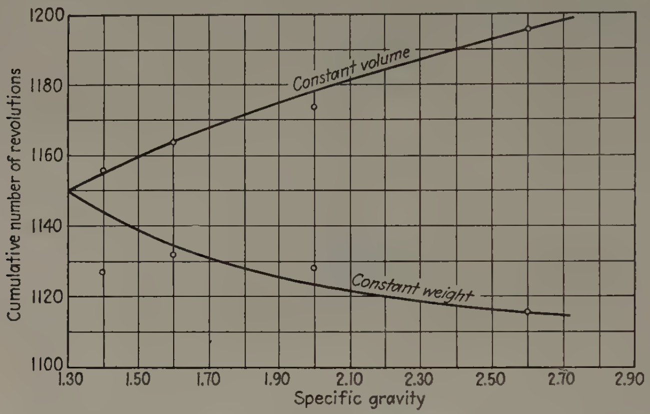

FIG. 1.—GRINDABILITY OF SPECIFIC GRAVITY FRACTIONS OF COAL *X* ($\frac{3}{8}$ TO $1\frac{1}{8}$ INCH) BY CONSTANT-WEIGHT AND CONSTANT-VOLUME METHODS.

In Table 8 the theoretical calculated values and actual volumes are shown for mixtures of 1.60 float and 1.60 sink for coal *Y*. From this table 500 grams of 1.60 float required 1179 revolutions and 500 grams of 1.60 sink required 925 revolutions, so a 50-50 mixture should theoretically require 0.5 × 1179 + 0.5 × 925 or 1052 revolutions, and the actual number by test was 1066. The 500 grams of this mixture occupied 605 c.c. volume so theoretically a 750 c.c. volume would require $^{750}\!/_{605} \times 1066$ or 1321 revolutions and by test required 1334.

TABLE 6.—*Calculated Number of Revolutions for a* 750 *C.c. Volume from Known Revolutions Required* (80 *Per Cent Minus* 200 *Mesh*) *for Another Volume, Coal X*

Specific Gravity	Volume C.c.[a]	Revolutions Required[b]	Revolutions for 750 C.c. Volume		Variation, Per Cent
			Calculated	Actual[b]	
1.30 float	750	1150	1150	1150	
1.30–1.40	730	1121	1152	1158	+0.5
1.40–1.60	650	1187	1370	1400	+2.2
1.60–2.00	560	1109	1485	1445	−2.7
2.00–2.60	400	861	1614	1603	−0.6
2.60 sink	340	700	1544	1505	−2.5
Head	710	1135	1200	1215	+1.2

[a] From Table 1.
[b] From Table 5.

Fig. 2 shows the close relationship between theoretical and actual values when "number of revolutions" is plotted against "1.60 float-and-sink mixtures." These curves show clearly that by the tentative standard ball-mill method the number of revolutions required to grind 80 per cent minus 200 mesh decreases as the amount of 1.60 sink increases. However, the volume method shows an increase in the number of revolutions required as the amount of 1.60 sink increases.

TABLE 7.—*Calculated Cumulative Number of Revolutions for Specific Gravity Fractions of* 750 *C.c. Volume from Known Revolutions Required for Another Volume, Coal X*

Specific Gravity	Cumulative Number Revolutions	
	Calculated	Actual
1.30 float	1150	1150
1.30–1.40	1152	1156
1.40–1.60	1159	1164
1.60–2.00	1170	1174
2.00–2.60	1194	1196
2.60 sink	1196	1199

TABLE 8.—*Grindability Values* (80 *Per Cent Minus* 200 *Mesh*) *of Mixtures* 1.60 *Float and* 1.60 *Sink, Coal Y*

Mixture, Per Cent Wt.		Volume, C.c.	500-gram Weight		750 C.c. Volume	
			Number Revolutions		Number Revolutions	
1.60 Float	1.60 Sink		Calculated	Actual	Calculated	Actual
100	0	740		1179	1195	1189
95	5	730	1166	1175	1207	1208
90	10	720	1155	1174	1223	1230
85	15	700	1140	1162	1245	1240[a]
80	20	690	1128	1160	1261	1258
75	25	670	1115	1145	1282	1274
50	50	605	1052	1066	1321	1334
25	75	530	989	994	1406	1424
0	100	470		925	1475	1502[a]

[a] Checked.

SHORTENING THE TENTATIVE STANDARD BALL-MILL METHOD TO THREE CYCLES

In the tentative standard ball-mill method, one of the main objections is the time required to run a test. Although the number of tests conducted to date is too limited to warrant a flat statement regarding the suggested shortening, nevertheless there is evidence in the data from 100 tests listed under Table 9 to justify some consideration for reducing the time involved.

The results of 100 grindability tests by the tentative standard ball-mill method were interpolated at 10 per cent values, ranging from 10 to 80 per cent. The number of revolutions required to produce 80 per cent of the sample as minus 200 mesh were then calculated when the number of revolutions to produce 10, 20, 30, etc., per cent of minus 200-mesh material was known. Thus the figures at the 10 per cent value would represent the number of revolutions required to produce 50 grams of minus 200-mesh material from a 500-gram sample multiplied by 8 to place all results on an 80 per cent basis, which is the method of reporting the final results on a full grindability test. In some cases the 20 per cent value checked the true value most closely; in others this was the 30 per cent value. However, Table 9, showing the distribution of the limits of error and the probable error expressed as per cent error, indicates that the values at 30 per cent agree more closely with the true value than any other.

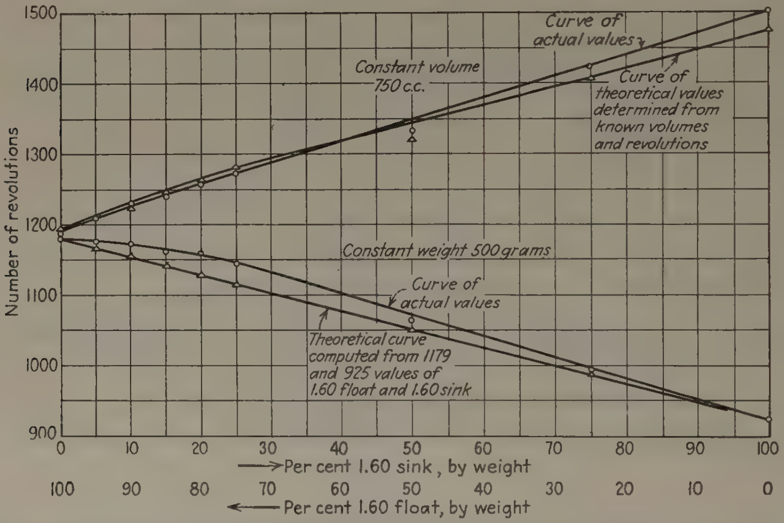

FIG. 2.—GRINDABILITY OF MIXTURES OF 1.60 FLOAT AND 1.60 SINK OF COAL *Y* (⅜ TO 1⅛ INCH) BY CONSTANT-WEIGHT AND CONSTANT-VOLUME METHODS.

A true grindability curve follows a straight-line function up to about 50 to 60 per cent. It then tapers, owing to the change in the physical characteristics of the material that accumulates and is the last to be ground. In calculating the number of revolutions at 80 per cent from the number of known revolutions at 10, 20 per cent, etc., it is apparent that a correction should be applied in order to change the actual conditions to a straight-line function. As stated previously, the number of revolutions yielding 30 per cent minus 200-mesh were used in the calculations in preference to those of other percentage yields because closer agreements between the calculated and the actual number of revolutions at 80 per cent could be obtained. The correction factor applied under these conditions was calculated as 1.5 per cent of the value calculated at 80 per cent. The magnitude of the correction factor was determined as the average difference in the calculated number of revolutions at 80 per cent from the known number of revolutions at 30 per cent and the actual number of revolutions at 80 per cent from the 100 tests. When this correction factor is applied approximately 50 per cent of the values obtained fall below and 50 per cent above the true values, whereas, before the factor was applied, about 80 per cent of the calculated results fell below and the others above the true values. It will be seen from Table 9 that when the results at 30 per cent and those at 30 per cent corrected for 1.5 per cent are compared a lower probable error will result, and although there is very little difference in the ±1.0 and ±2.0 per cent limits of error, it becomes more pronounced in the wider ranges.

TABLE 9.—*Distribution of Deviations from the "As-run" Value at 80 Per Cent within Various Limits*

Limit of Error, Per Cent	100 Tests, Per Cent				76 Tests Greater than 1000 Revolutions 30 + 1.0 Per Cent Correction	100 Tests, Per Cent			
	10	20	30	30 + 1.5 Correction		40	50	60	70
± 1.0	11	19	33	35	47	13			6
± 2.0	18	44	57	57	76	33	17	12	55
± 3.0	30	62	70	82	94	58	45	49	79
± 4.0	41	74	82	89	100	74	72	74	100
± 5.0	54	87	91	94		84	87	88	
± 6.0	69	91	91	97		90	91	100	
± 7.0	78	94	97	99		94	96		
± 8.0	86	99	99	100		97	99		
± 9.0	89	100	99			99	100		
±10.0	93		100			100			
±11.0	98								
±12.0	98								
±13.0	100								
Probable error per cent......	4.3	2.5	2.0	1.8	1.0	2.8	3.2	3.1	2.1
Distribution									
+80 value....			20	54	49				
−80 value....			80	46	51				

It was found in the 100 tests that the coals having the greatest discrepancy at the 30 per cent value fell below 1000 revolutions. There were 76 samples that have a grindability greater than 1000 revolutions and 24 samples less than 1000 revolutions. The average error between the 30 per cent value calculated to 80 per cent and the actual value at 80 per cent is 1 per cent for the 76 samples. If this corrected factor is applied to the 76 samples, the probable error is reduced to 1 per cent and 94 per cent of the samples fall within ±3.0 per cent error.

Therefore it would seem that a three-cycle run would be sufficient to obtain comparative results, especially on coals requiring 1000 or more revolutions. A correction factor of 1 per cent would be applied to the calculated result. However, in dealing with an unknown coal or one with a grindability index greater than 50 on the 50,000 index scale, it would be advisable to run a full eight-cycle test.

PREPARATION OF SAMPLES

In many laboratories the standard equipment does not include a laboratory-type double-roll crusher but consists of crushers and pulverizers of other types. Yancey, Furse and Blackburn[1] have shown that there can be a very large variation in the size composition of the minus 10-mesh sample without materially affecting the final results if the proper care is taken to avoid overcrushing. If the minus 4-mesh sample is reduced to minus 10 mesh in four to five stages as recommended by the American Society for Testing Materials, the type of machine used does not seem to play an important role. The data in Tables 10 to 12 give the results from samples prepared by stage crushing in rolls and a disk

TABLE 10.—*Size Composition of Minus 10-mesh Material Prepared on Parallel Samples by Using Rolls and McCool Pulverizer, and the Resulting Number of Revolutions (Tyler Screen Scale)*

Size, Mesh	0 to 3/8 In.		3/8 to 1 1/8 In.	
	Rolls	McCool Pulverizer	Rolls	McCool Pulverizer
10– 14	22.3	19.6	20.3	23.1
14– 28	37.2	35.8	40.9	41.0
28– 48	18.9	19.2	19.9	17.9
48–100	10.6	11.7	10.1	9.4
100–200	5.0	5.9	4.7	4.3
–200	6.0	7.8	4.1	4.3
Number of revolutions required, 80 per cent minus 200 mesh.........	1091	1085	1145	1152

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[1] H. F. Yancey, O. L. Furse and R. A. Blackburn: Estimation of the Grindability of Coal. *Trans.* A.I.M.E. (1934) **108,** 267.

pulverizer (McCool). The sample of ⅜ to 1⅛ in. was reduced to minus 4 mesh in a 00 Sturtevant crusher before being stage-crushed in a McCool pulverizer.

In Table 12, comparing the screen analyses of samples prepared to minus 10 mesh by stage-crushing in rolls and crushing to minus 4 mesh in a 00 Sturtevant and then stage-crushing to minus 10 mesh in a McCool pulverizer, there is a very close agreement. The slight difference is in favor of the McCool pulverizer, which produces less fines and consequently causes less overcrushing. However, the variation in these screen analyses should have no effect on the grindability results.

TABLE 11.—*Size Composition of Minus* 10-*mesh Material Prepared on Parallel Samples by Stage-crushing in Rolls by Pittsburgh Coal Company and U.S. Bureau of Mines and Resulting Number of Revolutions* (*Tyler Screen Scale*)

Size Mesh	Sample *A*		Sample *B*	
	Pittsburgh Coal Co.	Bureau of Mines	Pittsburgh Coal Co.	Bureau of Mines
10– 14	19.1	36.3	18.7	37.6
14– 28	38.7	35.3	40.3	33.9
28– 48	21.9	14.1	20.6	13.6
48–100	11.6	7.9	10.1	7.8
100–200	4.5	3.6	4.9	3.1
–200	4.2	2.8	5.4	4.0
Number of revolutions required 80 per cent minus 200 mesh.........	1390	1378	982	981

TABLE 12.—*Size Composition of Minus* 10-*mesh Samples Prepared by Rolls, and* 00 *Sturtevant and McCool Pulverizer* (*Tyler Screen Scale*)

Size Mesh	Average Screen Analyses, 17 Samples Prepared by Rolls	Average Screen Analyses, 8 Samples Prepared by 00 Sturtevant and McCool
10– 14	20.5	24.3
14– 28	39.3	40.6
28– 48	20.3	17.2
48–100	10.3	9.2
100–200	4.7	4.5
–200	4.9	4.2

Since the results in Tables 10 and 11 seem to show that there can be a wide variation in the size composition of the same sample as prepared by the same method in different laboratories or different methods in the same laboratory without affecting the grindability results materially, it is thought that an option in the choice of manner of preparing the sample should be incorporated in the procedure. The present status of preparing samples would eliminate any method other than roll crushers as not being standard, although no appreciable deviations are introduced into the final results. In fact, it would seem safe to include, under the standard method of preparation of the minus 4-mesh sample to minus 10 mesh, any crushing method, as long as the reduction is done in four or five stages and the minus 10-mesh material removed between crushing stages.

Screening of Minus 200-mesh Material

Under the test procedure given in Proposed Tentative Method of Test for Grindability of Coal by the Ball Mill Method, by A.S.T.M. Committee D-5 (1935 Report), the minus 200-mesh is removed per cycle by hand screening on a 74-micron (No. 200) sieve, 10 in. or more in diameter, until not more than 0.5 gram passes in one minute of screening. We do not understand why the use of a Ro-Tap machine, which uses 8-in. sieves, is excluded. It is our opinion that mechanical screening should be permitted as part of a standard procedure because most commercial laboratories are equipped with this type of screening device. We also believe vibration screening gives more consistent results than hand screening. The objection to degradation by mechanical screening is irrelevant, it has been found that at the end of the second cycle of an eight-cycle test, practically all of the material is minus 28-mesh, which size we believe is not broken down appreciably by any mechanical screening operation.

The screening operation in our investigation has been done in a nest of screens, 8 in. in diameter, on a W. S. Tyler Ro-Tap machine. For removing the minus 200-mesh material from the head sample and the first two cycles, the screens used were 14, 28, 48, 100 and 200 mesh. After the second cycle, there was practically no plus 14-mesh material left, and therefore the 14-mesh screen was removed from the nest and a 60-mesh screen inserted between the 48 and 100-mesh screens. Considerable work was done on determining the rate of screening and it was found that during the last five minutes of Ro-Tap of a 15-min. period, the rate was 2 to 5 grams. With this in mind we adopted a 15-min. Ro-Tap period as a standard. In removing the material from the screens, any adhering to the sides and screen cloth, top and bottom, was collected separately and screened again for 1 to 3 min. on the 200-mesh screen. This amounted to about 1 gram. Checks were also run by taking the 100 to 200-mesh material and hand-screening it for 1 min. on a 200-mesh screen, with a recovery of 200 to 300 mg. of minus 200-mesh. This is considerably less than 0.5 gram. The test was run on coal from the Lower Kittanning seam, which is very sticky by nature and difficult to screen.

From our screening investigation we feel certain that the minus 200-mesh material is thoroughly removed by Ro-Tap screening when the rate of screening is 5 grams or less during the last 5 min. of screening of a

15-min. Ro-Tap period. This screening limit was also checked against wet screening with very close checks. Even in screening samples of 2.60-sink material where 1100 grams was used on a volume test, a 15-min. Ro-Tap period was found to be sufficient to screen to that limit. The Hardgrove Standard Sample (Table 14) is a severe test on any screening operation, as this coal is very "sticky" in nature and extreme care must be exercised to keep the resulting losses to a maximum of 3 grams. This "stickiness" manifests itself in a tendency to cling to the balls and

TABLE 13.—*Tests Made by U.S. Bureau of Mines Using Hand Screening and Pittsburgh Coal Company Using Ro-tap Screening for 15-minute Period on Same Coal*
Minus 200-mesh Removed per Cycle, Grams

Cycle No.	Total Revolutions	Pittsburgh Coal Company			U.S. Bureau of Mines
		Test 1	Test 2	Average	Test 1
COAL A					
1	200	56	55	55	53
2	400	59	60	59	59
3	600	60	63	61	61
4	800	60	60	60	59
5	1000	60	57	59	60
6	1200	56	56	56	58
7	1400	53	49	51	49
8	1600	38	40	39	41
Oversize..................		58	60	59	60
Revolutions required, 80 per cent minus 200 mesh......		1390	1400	1395	1405
COAL A					
1	175	48			47
2	350	53			55
3	525	53			55
4	700	54			57
5	875	54			53
6	1050	53			53
7	1225	48			47
8	1400	43			46
9	1575	36			34
Oversize..................		58			53
Revolutions required, 80 per cent minus 200 mesh......		1380			1351

sides of the mill and in the screening operation tends to adhere to the sides of the screens.

TABLE 14.—*Tests Made by U.S. Bureau of Mines Using Hand Screening and Pittsburgh Coal Company Using Ro-Tap Screening for 15-minute Period on the Same Coals*

Cycle No.	Total Revolutions	Minus 200-mesh Removed per Cycle, Grams							
		Pittsburgh Coal Company					U.S. Bureau of Mines		
		Test 1	Test 2	Test 3	Test 4	Average	Test 1	Test 2	Average
COAL B									
1	125	50	52	50	52	51	52	50	51
2	250	53	52	51	53	52	52	54	53
3	375	55	54	55	55	55	54	55	55
4	500	55	55	56	55	55	57	57	57
5	625	54	55	55	54	54	55	54	54
6	750	50	50	51	53	51	51	50	50
7	875	47	48	48	47	47	46	48	47
8	1000	41	41	42	40	41	39	38	39
Oversize..............		95	93	92	91	94	94	94	94
Revolutions required, 80 per cent −200 mesh...		985	979	976	973	978	981	980	980
HARDGROVE STANDARD SAMPLE									
1	75	51	52	50	52	51	49		
2	150	54	53	55	53	54	51		
3	225	54	55	56	56	55	54		
4	300	52	53	54	54	53	54		
5	375	51	52	52	52	52	53		
6	450	45	46	50	49	47	47		
7	525	42	46	45	46	45	45		
8	600	40	39	38	38	39	40		
9	675	34	32			33	34		
Oversize..............		77	72	100	100	71	73		
Revolutions required, 80 per cent −200 mesh...		624	610	600	600	609	615		

Comparison of Limits of Screening between Ro-Tap and Wet-screening by Hand

Purpose.—The purpose of the following test was to determine the effect on the grindability caused by the addition of ¾ gal. of oil per ton of 0 to ⅜-in. coal. The oil was added for the purpose of "fixing" the dust. However, the test also will show the thoroughness of screening by the Ro-Tap method and the "cushioning effect" in the ball mill, by not removing the minus 200-mesh as a 10 per cent cycle operation.

Table 15.—*Test Showing Rate of Screening of Coal A at 5-minute Intervals of 15-minute Ro-Tap Period*

Cycle No.	Actual Weight Minus 200-mesh						U.S. Bureau Mines Test
	First 5 Min.	Second 5 Min.	Third 5 Min.	Total 15 Min.	Loss	15 Min. and Loss	
1	41	7	5	53	3	56	53
2	44	8	4	56	3	59	59
3	40	12	5	57	3	60	61
4	40	12	5	57	3	60	59
5	41	12	5	58	2	60	60
6	40	9	4	53	3	56	58
7	39	6	5	50	3	53	49
8	30	6	2	38	0	38	41
Oversize................						58	60
Revolutions required, 80 per cent −200 mesh....						1390	1405

Summary of Results.—The addition of ¾ gal. of oil per ton of 0 to ⅜-in. coal has no effect on the grindability and produced no "sticking" to the balls or mill. Screening by Ro-Tap to the limit of 5 grams or less for a 5-min. screening period checks very closely with wet screening by hand. For a cycle of 1000 revolutions 56 per cent, or 280 grams, is reduced to minus 200 mesh whereas a standard eight-cycle 10 per cent removal per cycle test on the untreated "regular sample" shows that at the end of 1000 revolutions 74.6 per cent, or 373 grams, is removed as minus 200-mesh.

Method of Test.—It was impracticable to use the U. S. Bureau of Mines Proposed Ball Mill Method on the oil-treated sample because of the difficulty of screening out the minus 200-mesh. A test was adopted in which two 500-gram samples of 10 to 200 mesh mine A 0 to ⅜-in., one oil-treated and the other a regular sample, were subjected to 1000 revolutions in the ball mill and the minus 200-mesh removed by screening on the Ro-Tap and wet screening in gasoline. Both samples were subject to the same treatment and the samples were screened to the same limits of screening. The minus 200-mesh was removed from the minus 10-mesh before grinding, by Ro-Tap. There may be a small disparity at this point, as no doubt all of the minus 200-mesh was not removed from the oil-treated sample.

After the samples were removed from the mill they were both given a 15-min. Ro-Tap. Because of the small amount of the minus 200-mesh removed from the oil-treated sample it was immediately wet-screened in gasoline; the plus 200-mesh dried and the minus 200-mesh taken as difference. The regular sample was additionally Ro-Tapped in 5-min. runs after the initial 15-min. Ro-Tap until the rate of screening was one gram per minute or less. The plus 200-mesh was then wet-screened in gasoline. Thereafter both samples, the oil-treated and the regular plus 200-mesh were Ro-Tapped for 5-min. intervals until the rate of screening was one gram per minute or less. In both cases the minus 200-mesh removed by wet screening was taken as the difference between the total sample and the plus 200-mesh after screening.

TABLE 16.—*Screen Analyses of Feed to Mill*

Size	Oil-treated Sample		Regular Sample	
	−10 Mesh Per Cent Wt.	10 to 200 Mesh Per Cent Wt.	−10 Mesh Per Cent Wt.	10 to 200 Mesh Per Cent Wt.
10– 14	22.7	23.4	19.5	21.2
14– 28	37.2	38.5	36.0	39.2
28– 48	19.0	19.6	19.1	20.8
48–100	12.6	13.0	11.6	12.6
100–200	5.5	5.7	5.7	6.2
Minus 200	3.0		8.1	

TABLE 17.—*Minus* 200-*mesh Removed by Screening after* 1000 *Revolutions*

	Oil-treated Sample			Regular Sample		
	+200 Mesh Grams	−200 Mesh		+200 Mesh Grams	−200 Mesh	
		Grams	Cum. Grams		Grams	Cum. Grams
15-min. Ro-Tap	466	34	34	336	164	164
5-min. Ro-Tap				274	62	226
5-min. Ro-Tap				257	17	243
5-min. Ro-Tap				248	9	252
5-min. Ro-Tap				242	6	258
5-min. Ro-Tap				237	5	263
Wet screen in gasoline	238	228[a]	262	231	6[a]	269
5-min. Ro-Tap	222	16	278	223	8	277
5-min. Ro-Tap	218	4	282	220	3	280
Per cent minus 200-mesh		56.4			56.0	

[a] By difference.

Discussion of Results

From the data given it will be seen that 56 per cent, or 280 grams, are removed by the one 1000-revolution cycle. A standard eight-cycle test on the same "regular sample" shows that at the end of 1000 revolutions 74.6 per cent or 373 grams is removed as minus 200-mesh.

TABLE 18.—*Tentative Standard Test on Regular Sample*[a]

Cycle	Total Revolutions	Minus 200-mesh		
		Weight Grams	Per Cent Weight	Cum. Per Cent Wt.
1	150	55	11.0	11.0
2	300	53	10.6	21.6
3	450	60	12.0	33.6
4	600	57	11.4	45.0
5	750	60	12.0	57.0
6	900	54	10.8	67.8
7	1050	51	10.2	78.0
8	1200	43	8.6	86.6
Oversize.........		67	13.4	100.0

[a] Calculated cumulative per cent weight at 1000 revolutions = 74.6 per cent or 373 grams.

REPORTING GRINDABILITY RESULTS—INVESTIGATION IN GRINDABILITY OF COAL SHOWING THE HIGHEST GRINDABILITY INDEX

Purpose and Summary of Results

To show the effect of different number of revolutions per cycle; the effect by leaching sample in gasoline before grindability test; the effect of heating sample before grindability test. The summary of results is given in Table 19.

TABLE 19.—*Grindability of Coal*

Test No.	Description	80 Per Cent through 200 Mesh	
		Total Revolutions	Grindability Index 50,000
1	Untreated, 75 revolutions per cycle	551	90.8
2	Untreated, 60 revolutions per cycle	540	92.6
3	Gasoline leached for 2 hr.[a]	568	88.1
4	Heated at 150° C. for 3 hr.[a]	570	87.7
Average		557	89.8

[a] 60 revolutions per cycle.

Sample and Treatment

The sample was a Lower Kittanning seam slack, from Cambria County, Pennsylvania, recognized as being the easiest coal to grind in all grindability studies conducted in this and other laboratories. It was prepared to minus 10-mesh by stage crushing in the McCool pulverizer. Grindability was determined by the tentative standard ball-mill method.

Tests were run, using 60 and 75 revolutions per cycle. Because these coals, which have a high grindability index, are of a "sticky" nature tending to adhere to the balls and mill, tests were also run at 60 revolutions per cycle on one sample that had been leached in motor gasoline for 2 hr. and one that had been heated in a container for 3 hr. at 150° C. The minus 200-mesh was removed before the samples were leached and heated and screened again after these treatments to remove any minus 200-mesh material caused by breakage in handling and treating.

The minus 200-mesh was removed by screening for 15 min. on a W. S. Tyler Ro-Tap machine in a nest of 8-in. sieves of 14, 28, 48, 100 and 200-mesh screens. After the second cycle the 14-mesh was removed and a 60-mesh inserted between the 48 and 100-mesh screens. The brushings from the sides and screen cloth were collected separately and the minus 200-mesh removed by a Ro-Tap screening of 1 or 2 min. The 100 to 200-mesh material was then given a hand screening until the rate of removal of the minus 200-mesh was less than 0.5 gram per minute, which never required over 1 min. of screening.

Discussion of Results

Although there is a maximum variation of 30 revolutions, the greatest deviation from the average will give a 3 per cent error. The difference between 75 and 60 revolutions per cycle produces no great effect on the final results, the variation being 11 revolutions. Both treated samples show a larger number of revolutions than the untreated samples. This may be due in part to the removal of certain components in the coal that aid in grinding. However, the two treated samples are well within a 3 per cent error of the average for the four tests.

Leaching in gasoline and heating both aid in reducing the difficulties of conducting a test. The tendency to stick to the balls and mill is greatly reduced in both cases. Heating the sample eliminates "stickiness" more than leaching. A shorter over-all operating time was required to run these tests because the balls and mill could be cleaned much more easily and the resulting losses were lower. The heat-dried sample was the easiest to run, required the shortest time and the resulting losses were the lowest.

This coal is the easiest to grind of all coal encountered in our investigation. Two other samples of Lower Kittanning slack also coming from this locality have been run and both required 560 revolutions.

TABLE 20.—*Tests on Lower Kittanning Seam Slack*

Cycle No.	75 Revolutions per Cycle				60 Revolutions per Cycle									
	Total Revolutions	Test 1			Total Revolutions	Test 2			Test 3			Test 4		
		Untreated				Untreated			Gasoline Leached for 2 Hr.			Heated at 150° C. for 3 Hr.		
		Through 200 Mesh				Through 200 Mesh								
		Weight Grams	Per Cent Weight	Cumulative Per Cent Weight		Weight Grams	Per Cent Weight	Cumulative Per Cent Weight	Weight Grams	Per Cent Weight	Cumulative Per Cent Weight	Weight Grams	Per Cent Weight	Cumulative Per Cent Weight
1	75	54	10.8	10.8	60	44	8.8	8.8	38	7.6	7.6	39	7.8	7.8
2	150	56	11.2	22.0	120	44	8.8	17.6	44	8.8	16.4	43	8.6	16.4
3	225	63	12.6	34.6	180	50	10.0	27.6	45	9.0	25.4	45	9.0	25.4
4	300	58	11.6	46.2	240	48	9.6	37.2	47	9.4	34.8	47	9.4	34.8
5	375	57	11.4	57.6	300	48	9.6	46.8	46	9.2	44.0	46	9.2	44.0
6	450	54	10.8	68.4	360	48	9.6	56.4	46	9.2	53.2	45	9.0	53.0
7	525	45	9.0	77.4	420	44	8.8	65.2	42	8.4	61.6	42	8.4	61.4
8	600	38	7.6	85.0	480	40	8.0	73.2	39	7.8	69.4	41	8.2	69.6
9					540	34	6.8	80.0	37	7.4	76.8	36	7.2	76.8
10					600				34	6.8	83.6	32	6.4	83.2
Oversize		75	15.0	100.0		100	20.0	100.0	82	16.4	100.0	84	16.8	100.0
Loss between ±200 mesh				16				11			9			5
Revolutions required 80 per cent −200 mesh				551				540			568			570

Screen Analyses, 10 to 200 Mesh

Size, mesh	Test 1 Per Cent Weight	Test 2 Per Cent Weight	Test 3 Per Cent Weight	Test 4 Per Cent Weight
10– 14	26.0	27.9	25.7	27.7
14– 28	40.0	39.4	39.7	39.3
28– 48	19.0	19.2	19.2	18.5
48–100	10.4	9.4	10.5	9.9
100–200	4.6	4.1	4.9	4.6
	100.0	100.0	100.0	100.0

It is our opinion that all results should be reported on an index scale of 50,000. This would mean that the grindability of a coal would be expressed as an index number obtained by dividing 50,000 by the average number of revolutions required to grind 80 per cent minus

TABLE 21.—*Checks on Same Samples Run in Same Laboratory*

TENTATIVE STANDARD BALL-MILL METHOD

Sample No.	Revolutions Required, 80 Per Cent Minus 200-mesh				
	Test 1	Test 2	Test 3	Test 4	Maximum Deviation, Per Cent
1	1441	1432	1437		0.6
2	1390[a]	1400[a]	1380[b]		1.4
3	1294	1283			0.9
4	1270	1261			0.7
5	1183	1175			0.7
6	1177	1197			1.7
7	1150	1150			0.0
8	1145[c]	1152[d]			0.6
9	1138	1139			0.1
10	1091[c]	1085[d]			0.5
11	1068[e]	1050[f]			1.7
12	985	979			0.6
13	868	855			1.5
14	616	624			1.3
15	624[g]	610[g]	600[h]	600[h]	3.9
16	584	590			1.0
17	551[i]	540[j]			2.0
Average.					1.1
VOLUME METHOD					
1	1150	1150			0.0
2	1600	1605			0.3
3	1503	1500			0.2
4	1231	1249			1.4
Average.					0.5
Total average.					1.0

[a] 200 revolutions per cycle.
[b] 175 revolutions per cycle.
[c] Staged-crushed in rolls.
[d] Staged-crushed in McCool.
[e] 125 revolutions per cycle.
[f] 135 revolutions per cycle.
[g] Run by Operator 1.
[h] Run by Operator 2.
[i] 75 revolutions per cycle.
[j] 60 revolutions per cycle.

200-mesh or Grindability Index $= \dfrac{50,000}{\text{Average number of revolutions}}$. Thus, for a Lower Kittanning slack the grindability index would be $\dfrac{50,000}{560}$, or 89.3. In using this scale, no grindability index figure greater than 100 would be obtained, and for comparative purposes index figures ranging from 0 to 100 are more suitable than indices greater than the 100 range.

PERMISSIBLE ERROR

Error on Duplicate Tests Made by Same Laboratory Shall Not Exceed Three Per Cent.—We believe that the limits of error allowed on duplicate tests in the same laboratory should be expressed on a percentage basis with the allowable variation of 3.0 per cent rather than on a revolution basis. The Tentative Standard Ball-mill Method now states (A.S.T.M. Committee D-5 Report on Coal and Coke for 1935) under Precision of Tests, that duplicate tests made in the same laboratory shall agree within 50 revolutions. If a check of 50 revolutions is considered as being sufficient for two tests on the same coal, this would mean a check of 4 per cent on a coal requiring 1250 and 8 per cent on a coal requiring 625 revolutions. It has been our experience that coals with a grindability index (50,000 scale) of 50 or under are very easy to check within a few revolutions and, although coals of a higher grindability index are harder to check, nevertheless they can be checked within very close limits.

ADOPTION OF ONE STANDARD METHOD

The ball-mill method and the Hardgrove method as adopted for tentative standards by the A.S.T.M. do not give results that can be accurately correlated. The reason, no doubt, lies in the difference of action in the two mills; the ball mill uses impact and abrasion, and the Hardgrove, pressure and abrasion. If more than one method is adopted as standard, the data obtained from any one of the methods should be comparative to the others. This, however, is not true with the methods mentioned and may cause confusion in the interpretation of the results obtained. It seems that the most practical way of obtaining grindability indices is by eliminating surface factors and fine screening because the former are inaccurate and the latter difficult and expensive.

CONCLUSIONS

1. The 500-gram weight of the 10 to 200-mesh material should be changed to approximately 750-c.c. volume.
2. It might be permissible to shorten the method to 30 per cent minus 200-mesh and to calculate the results obtained on an 80 per cent basis

whenever the characteristics of the coal in question are known especially if the coal has a grindability index of 50 or more.

3. The manner of preparing the sample to 1680 microns (minus 10-mesh, Tyler) should be optional in regard to type of crushing and screening equipment used, provided the method of preparation is followed.

4. The selection of hand or mechanical screening should be optional.

5. All grindability results should be reported on the 50,000 grindability index scale.

6. The error on duplicate tests made by the same laboratory should not exceed 3 per cent.

7. One method only should be chosen as a standard because the results obtained by the two methods, now tentative standards, are not comparable.

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[For discussion, see page 369]

Further Investigation of Methods for Estimating the Grindability of Coal

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(New York Meeting, February, 1936)

At the annual meeting of this Institute held two years ago a new method[1] of estimating the grindability of coal was described, based on experimental work carried on by the Bureau of Mines at its Northwest Experiment Station. Since that time, several organizations concerned with the testing or utilization of coal as pulverized fuel have adopted and used the method in determining the grindability of several hundred samples of coal. In addition, a series of comparative tests of this method and three others has been carried out in five laboratories under the auspices of Committee D-5 of the American Society for Testing Materials. As a result the Committee has recommended that the Bureau of Mines method, designated as the "ball-mill method," and the Hardgrove method[2], previously advanced, be accepted as tentative standards[3].

Since the ball-mill method was originally developed and published, an opportunity has been presented to apply it to a larger number of coals and to study intensively some of the principles upon which the method is based. This paper presents a résumé of these further studies and includes a comparison with the Hardgrove-machine method. Trial runs were also made with another procedure recently proposed—the C.I.T. roll-test method[4].

Principles of Methods

In the ball-mill method, the relative amounts of energy necessary to grind coals to the same fineness are determined by the number of revolutions of the mill required to reduce 80 per cent of the feed (500 grams of 10 to 200-mesh coal) fine enough to pass a 200-mesh screen. This is a

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[1] References are at the end of the paper.

somewhat finer size than that commonly used for pulverized-coal firing. The finished product is removed in increments of 10 per cent by stopping the mill and screening out the undersize at the end of each cycle. This prevents overgrinding, maintains a more constant size distribution in the subsieve material, and simulates the continuous removal practiced industrially.

In the Hardgrove-machine method, 50 grams of coal, sized between 14 and 28-mesh (Tyler) sieves, is ground in a special ring-and-ball machine for 60 revolutions. The resulting product, in which the amount of new surface is estimated from a screen analysis, is considerably coarser than a pulverized fuel.

The difference in the principles by which these two methods measure grindability is of particular importance. The ball-mill method is the only one so far proposed in which the relative amount of work required to grind coal to pulverized-fuel size is determined. By all other methods an equal amount of work is performed on each sample and the relative grindability obtained by estimating the new surface produced. Such procedures, although they may be reproducible, are subject to the limitation that by far the greater amount of surface is concentrated in the subsieve sizes; that is, the sizes that are finer than the finest sieves available. The approximation of surface in subsieve coal is an exacting and time-consuming task. Consequently, methods specifying a constant amount of work apply the same assumed mean size to the subsieve material from any and all coals. In order to test the validity of this assumed constant mean-size value, and to determine the effectiveness of stage removal in the ball-mill method, a study was made of the subsieve material obtained by testing three coals by both methods.

Study of Subsieve Material

The coals selected for these tests covered a wide range in grindability, as is shown in Table 1. Coal 1, from the Pocahontas No. 3 bed in West Virginia, is very easy to grind; coal 3, from the No. 6 bed in Illinois, is moderately difficult to grind, and coal 5, a Pennsylvania anthracite, is very hard to grind.

Determination of Subsieve Sizes

A number of methods are available for determining average particle size in subsieve material. Of these, sedimentation with calculation of particle size from Stokes' law and sedimentation or elutriation into sized fractions in combination with microscopic counting and measuring are often used. In this study both sedimentation and elutriation into size fractions were tried.

The sedimentation procedure and apparatus used were those of Andreasen,[5] as modified by Sloman and Barnhart,[4] who applied the method to the determination of the size distribution of subsieve coal from grindability tests, and concluded that the average particle size of coal passing a 300-mesh sieve is independent of the grinding characteristics of the coal or of the amount of grinding, within the limits to which a coal would be subjected in a grindability test.

TABLE 1.—*Grindabilities of Three Coals by Ball-mill and Hardgrove-machine Methods*

Coal No.	Actual		Relative	
	Ball Mill, Revolutions	Hardgrove, Per Cent	Ball Mill	Hardgrove
1	595	100.5	100	100
3	1166	60.1	51	60
5	1939	36.6	31	36

Sedimentation tests were made on the through 400-mesh material obtained from Hardgrove grindability trials on the three coals shown in Table 1. To minimize eddy currents within the sedimentation vessel, it was submerged in a constant-temperature water bath. Duplicate tests on all of the samples showed that the method was reproducible when the same lot of 95 per cent alcohol was used as a settling medium. However, the use of the alcohol recovered from previous tests by distillation gave a different size distribution on the same sample. Moreover, sedimentation as employed, with calculation of particle diameter from Stokes' law, modified for shape of particles, is obviously an indirect procedure. The American Society for Testing Materials has recommended a more direct method[6] in which the subsieve material is separated by elutriation or sedimentation into a series of closely sized fractions. These are weighed, and the particle sizes are determined by microscopic measurement.

The subsieve material obtained in the grindability tests was screen-sized with 325 and 400-mesh Tyler sieves, and the through 400-mesh portion was elutriated pneumatically with the Roller apparatus[7] into four or more closely sized fractions. A small amount of each fraction was then mounted on a slide, dispersed in xylene, and examined by projecting the image of the particles on a ground glass mounted above the microscope. This method of examination allows sharp focus on each particle and permits counting and measurement with a minimum of eye strain. If the material on the slide so prepared was dispersed satisfactorily, about 200 particles were measured in several fields; otherwise a new slide was prepared from the same elutriated fraction. The fields chosen were selected at random across the slide to eliminate any error that might be caused by using only one field.

The magnification used for a given fraction depended upon the size of the particles in that particular cut. Fractions of the finer sizes were magnified 500 times; the coarser fractions, 100 times. The average size was calculated by means of the weight mean formula $\frac{\Sigma nd^4}{\Sigma nd^3}$, in which n is the number of particles and d the diameter. From the average particle sizes of the fractions the weight mean of the composite was calculated by the equivalent form $\frac{\Sigma wd}{w}$, in which w is the weight of the fractions.

Comparison of Subsieve Sizes

Table 2 shows the results of the particle-size determinations on the portion of the subsieve coal that passes 400-mesh. The last two columns of the table show the average particle sizes of the total subsieve material. Two important conclusions may be drawn from these figures: First, in spite of the greater range in particle sizes occasioned by use of the 200-mesh sieve, the alternate grinding and screening of the ball-mill method produces subsieve material with only half the variation in average particle size of that produced in the Hardgrove method; and second, the average particle size is related to the grinding characteristics of the coal. This is particularly evident in the Hardgrove tests, in which coals 1, 3 and 5, with grindabilities of 100, 60 and 36, have mean sizes of 29, 32 and 37 microns, respectively. This second conclusion is not in agreement with that of Sloman and Barnhart, previously mentioned, to the effect that the average particle size is independent of the grinding characteristics of the coal or the amount of grinding.

TABLE 2.—*Average Particle Size of Subsieve Material Obtained from Grindability Tests*
Microns

Coal No.	Through 400-mesh		Through 200-mesh Ball Mill	Through 325-mesh Hardgrove
	Ball Mill	Hardgrove		
1	29	24	54	29
3	26	28	57	32
5	31	34	58	37

The data in Table 2 may be used to adjust the Hardgrove grindability values for variations in mean particle size of subsieve material. To do this, the particle sizes may be considered as relative and expressed in terms of coal 1 in the same way as are the grindability indexes. The particle sizes determined for coals 3 and 5 are 11 and 27 per cent, respectively, greater than that of coal 1. If Hardgrove's assumed value of 25.29 microns is increased by these percentages, the resulting sizes are in the same ratio as the determined values and can be used for calculating the adjusted surface factors to be applied for the subsieve fraction of each coal. These factors are 47.5 for coal 1, 42.8 for coal 3, and 37.4 for coal 5. The adjusted grindabilities obtained by the use of these factors are shown in Table 3. Since the mean particle size of the subsieve material produced by the ball-mill method is somewhat larger for the less grindable coals than for the more grindable, the ball-mill results also should be slightly lower for the harder coals. The close agreement between the adjusted Hardgrove values and the ball-mill results, as shown in Table 3, would not be materially affected by this change.

TABLE 3.—*Influence of Particle Size of Subsieve Material on Hardgrove Grindability Values*

Method	Coal No.		
	1	3	5
Ball mill	100	51	31
Hardgrove	100	60	36
Hardgrove, adjusted for particle size	100	53	31

EFFECT OF SELECTIVE GRINDING

Grindability is a composite property, obviously dependent on a group of specific properties such as cleavage, fracture, tenacity, hardness, and elasticity, which vary with the change in rank of coal, between coals of the same rank, and even between components of an individual coal. The variation of these properties between the components of many coals causes them to grind as heterogeneous mixtures of hard and soft material.

Several investigators[8,9] of the problems encountered in grinding have referred to the differential action of grinding machines if the feed consists of materials varying in grindability. If hard and soft materials are ground in the same operation the type of mill used determines to a great extent the relative reduction in size of each material. Some types give nearly the same reduction in size of hard and soft components; others show a marked differential grinding action. The degree to which such action takes place may be termed the selectivity of the mill in question. The nature and quantity of the components present in the feed determines the extent to which a given mill may selectively grind the less resistant portions. Selectivity influences the accuracy or correctness of a grindability procedure even though it does not influence the other prime requisite—its precision or reproducibility.

Ash Content and Selectivity

For a given coal, differences in ash contents of products obtained in grinding are undoubtedly due to differences in the relative grindability of the components with which the ash is associated. If the ash contents of all the cycle products or size fractions were equal, the sample might be assumed to have uniform grindability. To determine the relative selectivity existing in the ball-mill and Hardgrove methods, grindability tests were made on a series of six coals, and the ash contents of the cycle fractions from the ball-mill tests and the screen-size fractions from the Hardgrove tests were determined. Table 4 shows the source of the coals used and their relative grindabilities, which varied over a wide range.

TABLE 4.—*Comparison of Grindabilities of Six Coals by Ball-mill and Hardgrove-machine Methods*

Coal No.	Source			Actual			Relative	
				Ball Mill		Hard-grove	Ball Mill	Hard-grove
	State	Mine	Bed	Rev.[b]	Rate,[c] Lb. per Hour	Per Cent		
100[a]	Pa.	Jerome	Upper Kittanning	630	4.20	100	94	100
1	W. Va.	Lick Branch	Pocahontas No. 3	595	4.45	100.5	100	100
2	Pa.	Banning	Pittsburgh	1057	2.50	66.9	56	67
3	Ill.	No. 2	No. 6	1166	2.27	60.1	51	60
4	Pa.	Bear Valley	(Anthracite)	1342	1.97	50.9	44	51
5	Pa.	St. Nicholas	(Anthracite)	1939	1.36	36.6	31	36

[a] Hardgrove standard coal.

[b] Number of revolutions required to produce 80 per cent of material passing 200 mesh.

[c] Rate of grinding, pounds of product per hour, 80 per cent of which passes 200 mesh.

The ash contents of the fractions produced in each cycle in the ball-mill tests are given in Table 5. Table 6 shows the ash contents of the screen-size fractions obtained in the Hardgrove-machine method tests of these same coals. Inspection and comparison of Tables 5 and 6 show a greater difference in the ash contents of the fractions produced in the Hardgrove procedure than in those produced by the ball-mill method. A maximum difference in the Hardgrove tests occurred with coal 100, where the 14 to 28-mesh size carried 19.4 per cent of ash and the 200-mesh to 250-mesh size contained only 5.3 per cent. The greatest difference with the ball-mill method, also for coal 100, was from 15.8 per cent to 7.9 per cent ash. With both methods the difference in ash contents is greatest with the softest coals. The results in Table 6 for the Hardgrove tests show that the finer fractions are generally lower in ash than the coarser fractions. This method gives higher relative values than the ball-mill method, because it is susceptible to selectivity in a more marked degree. Only a relatively small portion of the sample is ground to the size actually used in powdered-coal firing, and the grinding effort required for the remainder of the sample is estimated from the work expended in pulverizing the portion that is the easiest to grind.

TABLE 5.—*Ash Contents of Finished Products, through 200-mesh, Produced by Each Cycle of the Ball-mill Method*

Cycle No.[a]	Coal No.					
	100	1	2	3	4	5
Feed	10.4[b]	5.3	7.0	14.3	9.4	9.2
1	8.5	4.8	5.9	14.7	9.4	10.9
2	7.9	4.7	5.8	14.2	9.1	10.2
3	8.0	4.7	5.9	13.9	9.1	9.8
4	8.1	4.8	6.1	13.8	9.0	9.5
5	8.5	4.9	6.4	13.9	9.1	9.2
6	9.0	5.1	6.8	13.8	9.2	8.9
7	9.8	5.3	7.4	13.9	9.3	8.8
8	10.7	5.6	8.1	14.1	9.6	8.5
9	11.8	5.9	9.1	14.6	10.0	8.2
10	13.6	6.3		15.6		
11	15.8					

[a] The number of revolutions in a cycle for each coal is as follows: No. 100 and No. 1, 75; No. 2 and No. 3, 150; No. 4, 175; No. 5, 250.

[b] All ash percentages on a moisture-free basis.

TABLE 6.—*Ash Contents of Screen-size Products from Tests by Hardgrove Method*

Size Mesh[a]	Coal No.					
	100	1	2	3	4	5
14 to 28	19.4[b]	9.8	10.2	21.2	13.3	9.8
28 to 60	9.8	5.7	7.4	13.1	10.6	9.5
60 to 100	7.0	4.5	6.2	10.6	9.2	9.2
100 to 150	6.1	4.2	5.8	10.7	8.6	9.3
150 to 200	6.0	4.1	5.7	10.7	8.5	9.6
200 to 250	5.3	3.7	5.3	10.1	8.4	9.7
250 to 300	5.9	4.0	5.6	13.1	10.1	10.5
−300	5.8	4.2	5.4	14.1	9.5	11.0

[a] Tyler standard sieves.

[b] All ash percentages on a moisture-free basis.

Selectivity in Mixtures

To present quantitative proof of this selectivity, a synthetic coal was prepared, consisting of a mixture of 50 per cent by weight of the coal least difficult to grind (No. 1) and 50 per cent of that most difficult to grind (No. 5). The grindability of the mixture should be the average of the grindabilities of the individual coals. Table 7 gives the results of grinding the synthetic coal by the ball-mill method in comparison with those obtained on each coal separately. To reduce 80 per cent of the mixture to pass a 200-mesh sieve required 1137 revolutions; the average for coals 1 and 5 was 1267 revolutions. Expressed in relative grindability indexes in terms of coal 1, the results are 52 for the actual test of the mixture and 47 for the average of the two coals—a difference of only 5 units.

The grindability of the mixture by the Hardgrove-machine method was 83 per cent—compared with 68 per cent—the mean of the grindabilities of coals 1 and 5. This is a difference of 15 units between actual and calculated grindabilities of the mixture for the Hardgrove method—compared with a difference of 5 units for the ball-mill method, due to the more selective grinding of the former method.

TABLE 7.—*Ball-mill Tests of a Mixture of Coals* 1 *and* 5, 50 *Per Cent by Weight*

Coal No.	Through 200-mesh, Per Cent	Grindability, Revolutions	
		Determined	Calculated
1	80	595	
	90	721	
	95	793	
5	80	1935	
	90	2216	
	95	2358	
Mixture of 1 and 5. .	80	1137	1267
	90	1389	1469
	95	1537	1576

Coal 1 is a low-volatile bituminous coal; coal 5 is an anthracite. Although in practice these coals may never be ground together for pulverized-fuel firing, they exhibit such wide differences in grindabilities that they are particularly suited to use for testing the accuracy and selectivity of the two methods. Since they are widely different in volatile-matter content—coal 1 has 20.0 per cent and coal 5 has 3.9 per cent on a moisture-and-ash-free basis—the proportion of each coal in all of the cycle and screen-size products may be determined, and thus the relative selectivity of the two methods may be examined and compared. Table 8 shows the cumulative percentages of coals 1 and 5 present in the products of the ball-mill tests as grinding proceeded. The first 10 per cent increment of finished product removed from the mill during the test consisted of 80 per cent of coal 1 and 20 per cent of coal 5, instead of 50 per cent of each. This shows that the softer coal was crushed at a more rapid rate than the harder coal. However, as grinding proceeded less of coal 1 and more of coal 5 was ground, until at 80 per cent, the usual end point of the ball-mill method, the finished product passing a 200-mesh sieve consisted of 58 per cent of coal 1 and 42 per cent of coal 5. If selective grinding were not done in the mill the total finished product would, of course, contain 50 per cent of each coal. Because coal 1 exceeds coal 5 in amount at the 80 per cent end point, the method gives a relative grindability 5 units higher for the mixture than for the mean of the individual components. If grinding is continued until 90 per cent or more of the sample is ground to pass a 200-mesh sieve, the actual and the calculated grindabilities agree almost exactly.

Table 8.—*Composition of Subsieve Coal Produced in Ball-mill Test on a Mixture of Coals* 1 *and* 5, 50 *Per Cent by Weight*

Through 200-mesh, Per Cent	Composition, Per Cent		Through 200-mesh, Per Cent	Composition, Per Cent	
	Coal 1	Coal 5		Coal 1	Coal 5
10	80	20	60	68	32
20	79	21	70	63	37
30	77	23	80	58	42
40	75	25	90	52	48
50	71	29	100	47	53

The composition of each of the screen-size fractions from Hardgrove-machine tests of the 50-per-cent-by-weight mixture of coals 1 and 5 was

Table 9.—*Calculated Composition of Size Fractions from Hardgrove Test on an Equal-weight Mixture of Coals* 1 *and* 5

Size, Mesh	Weight, Per Cent	Composition, Per Cent		Per Cent of Total Surface
		Coal 1	Coal 5	
14 to 28	31.0	9	91	3.9
28 to 60	27.4	42	58	6.8
60 to 100	9.6	76	24	6.0
100 to 150	5.4	82	18	4.1
150 to 200	5.4	86	14	6.1
200 to 250	2.4	86	14	4.6
250 to 300	0.4	89	11	5.5
−300	10.4	93	7	63.0

determined in a similar manner. Table 9 shows these results. As will be observed, none of the size fractions consist of a 50 per cent mixture of each of the two coals. The coarsest fraction consists of 9 per cent of coal 1 and 91 per cent of coal 5. The finest fraction contains 93 per cent of coal 1 and 7 per cent of coal 5.

Perhaps the best way of visualizing the difference between the two methods from the standpoint of selectivity is to calculate the percentage reduction in average size of each coal. This calculation showed that at 80 per cent through 200 mesh, the usual end point of the ball-mill method, coal 1 had undergone a reduction in average particle size of 94 per cent and coal 5 of 90 per cent. At 90 per cent through 200 mesh the reductions were 94 and 93 per cent, respectively. In the Hardgrove tests of the mixtures the average particle-size reduction amounted to 77 per cent for coal 1 and 27 per cent for coal 5. This comparison shows that selectivity, even in the extreme instance presented here, does not affect the accuracy of the ball-mill method, provided the sample is ground until 90 per cent passes 200 mesh. With the Hardgrove method, under similar conditions, accurate results cannot be expected.

The work with mixtures of hard and soft coals has demonstrated that both methods use mills that grind selectively. The advantage of the ball-mill method in this respect lies in the fact that so large a proportion of the material in a given sample, regardless of the variation in the grindability of the components, is ground to such a fine size that the hard material cannot remain unground. On the other hand, with the Hardgrove method only the softest components are ground to the finer sizes, where they necessarily reflect inordinately large surface values and hence grindability values.

The mixture of coals 1 and 5 probably allows maximum opportunity for selective grinding. When the ratio of grindabilities of the components of a coal is less than that existing in the mixture tested (3 to 1), and when the difficultly grindable portion is less than 50 per cent in amount, the possibility for selectivity is considerably less. However, in many of the softer coals the higher-gravity fractions occur in considerable quantities and are from one-half to one-third as grindable as the low-gravity fractions. With these coals the Hardgrove method surely must suffer from selective grinding; very likely the difference of about 10 units between the values given by the two methods for such coals can be attributed, at least in large part, to selectivity.

VOLUME OF MILL FEED

The efficiency of any grinding mill depends on the amount of material being ground. The two tentative methods proposed specify that they are intended for use in determining the grindability of coal. When another material with a widely different density is placed in the mill the volume occupied by the charge necessarily varies. Consequently the effectiveness of the mill also varies. With extreme differences in the bulk density of the charge, the grindability values obtained are not comparable. Under such conditions, grindability tests should be made upon equal numbers of particles per mill charge; that is, upon equal volumes of charge rather than upon equal weights. Maxon, Cadena, and Bond[8] point out that the volumes of the charges in full-size mills grinding ores of different specific gravities are more nearly equal than their weights.

Coals 1 and 5, previously tested on an equal-weight basis, are considerably different in density. Therefore, tests were made upon them to determine the effect of using equal volumes rather than equal weights of mill charges.

The selection of a standard volume of coal for the tests of equal volumes in the ball mill was based upon the bulk density of a bituminous coal from the Pittsburgh bed. The volume desired was one sufficiently large to hold 500 grams—the usual weight-basis sample. A section was cut from a Bureau of Mines coal-sample can, and a flat bottom was soldered on one end. This container, 6 in. long and 3⅛ in. in inside diameter, has a volume of 755 c.c. and delivers about 750 c.c. It was filled with 10 to 200-mesh coal, from a 6¾-in. 60° glass funnel, so supported that its tip, which had a bore of 5⁄16 in., was held 1½ in. above the top of the can. After the can had been filled to overflowing, the coal was leveled off by one sweep of a spatula, care being taken to avoid any jar. The volume of 750 c. c. obtained in this manner gave 496 grams of coal 2; the amount may be duplicated to within 2 grams.

Table 10 shows the results of the grinding tests made upon equal volumes of five coals by the ball-mill and the Hardgrove methods compared with the previous tests made upon equal weights of the same samples. Coals 1, 2 and 3 do not show significant differences between the volume and weight bases, because the standard volume of these coals weighs about 500 grams. However, coals 4 and 5, one a semian-

TABLE 10.—*Relative Grindabilities of Equal Volumes and Equal Weights of Five Coals Based on Coal* 1 *as* 100 *Per Cent*

Coal No.	Weight of Std. Volume, Grams	Relative Grindability			
		Ball Mill		Hardgrove	
		Volume	Weight	Volume	Weight
1	513	100	100	100	100
2	496	56	56	68	67
3	516	49	51	58	60
4	540	41	44	47	51
5	633	25	31	28	36

thracite and the other an anthracite, do show significant differences. For example, coal 5, which had a grindability of 31 by the ball-mill method on an equal-weight basis, has a grindability of 25 on the equal-volume basis.

Table 11 shows a similar series of tests contrasting the equal-volume and equal-weight bases on the specific-gravity fractions of coal from the No. 7 bed, Wilkeson, Wash. By both methods on both bases, the grindabilities of the specific-gravity fractions decrease with increase in specific gravity, because the coal itself is very soft, friable, and easy to grind, while the impurities higher in gravity are harder to grind. This table shows that the volume tests exhibit a considerably greater range than the weight tests. By the ball-mill method the range shown is from 100 to 30 in the volume tests, and from 100 to 47 in the weight tests. Significant differences are shown by both methods on the three heavier fractions. When the weight of the standard volume exceeds 500 grams by as much as do the weights of coals 4 and 5, as shown in Table 10, the volume procedure is considered to reflect the relative differences in grindability more accurately than does the weight procedure.

TABLE 11.—*Relative Grindabilities of Equal Volumes and Weights of Wilkeson No. 7 Float-and-sink Fractions, Based on Float* 1.30 *as* 100 *Per Cent*

Specific Gravity	Wt. of Std. Volume, Grams	Relative Grindability			
		Ball Mill		Hardgrove	
		Volume	Weight	Volume	Weight
Under 1.30	503	100	100	100	100
1.30 to 1.40	522	84	87	85	86
1.40 to 1.50	560	62	68	65	69
1.50 to 1.70	608	47	58	51	58
Over 1.70	780	30	47	35	51

Another matter of interest in Table 11 is the agreement of the ball-mill and the Hardgrove values. In all comparisons previously made the Hardgrove values are significantly higher than the ball-mill values. Such is not the case for the samples shown in Table 11, because the specific-gravity fractions on which the tests were made are more nearly homogeneous with respect to the grindability of the components than is the ordinary sample of coal. Hence there is no opportunity for selective grinding to be done in the Hardgrove machine. When hard and soft materials are not present in the same sample, the small amount of feed that is finely ground in the Hardgrove procedure reflects more accurately the relative grindability and gives values closely approximating those obtained by the ball-mill method, wherein a large proportion of the sample is ground to pulverized-fuel size.

RELATIONSHIP OF METHODS

Fig. 1 shows ball-mill grindability values plotted against Hardgrove values for a group of 16 coals, covering a wide range of grinding characteristics. The shape of the curve shows there is no simple relationship between the results obtained by the two methods. However, approximate conversion from one method to the other is made possible by use of the curve. The method of converting ball-mill values to Hardgrove values by multiplying the reciprocal of revolutions by 72,000 is obviously erroneous.

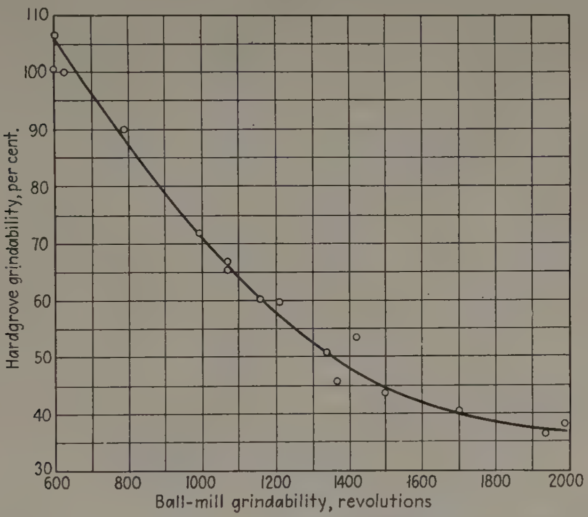

FIG. 1.—RELATION OF BALL-MILL AND HARDGROVE-MACHINE GRINDABILITY VALUES.

Investigation of C.I.T. Roll Test

The C.I.T. roll-test method[4] recently proposed possesses attractive features; tests are easily and rapidly performed. In brief, a sample of sized coal placed on a flat steel plate is crushed by rolling a heavy steel cylinder over it 10 times. The crushed product is then screen-sized, and the new surface produced is calculated from the screen analysis. In principle, the method may be characterized as belonging to the class of constant-work methods, such as the Hardgrove, and is subject to the same criticisms. As C. M. Smith[10] has said, "the enormous amount of new surface in finer sizes is so dominant and yet so indeterminate as to overshadow everything else, and we have the estimated behavior of a small percentage of the test material determining almost the entire result."

An apparatus was constructed, and the method was given a very careful trial. After about 30 tests had been made to secure experience with the method, it was applied to five different coals, which had been tested previously by the ball-mill and Hardgrove methods. These tests, made in quadruplicate, are shown in Table 12 in comparison with results obtained in the laboratory of the originators of the method and also with those obtained on the same coals by the ball-mill method. Despite rigid adherence to the published procedure, the data in Table 12 show that the method is not precise. The exceedingly small weight of sample used (20 grams) makes the method particularly susceptible to error due to the losses ordinarily encountered in making screen analyses. Additional work on the method, with more rigorous specification of procedure, may greatly improve the reproducibility.

TABLE 12.—*Relative Grindabilities of Five Coals by C.I.T. Roll-test Method*

Coal No.	Units	C.I.T. Method					Ball-mill Method
		Test 1	Test 2	Test 3	Test 4	Tests A[a]	
1	Surface	16684	16720	18745	17788		
	Grindability	100	100	100	100	100	100
2	Surface	11135	10234	11390	12073		
	Grindability	67	61	62	66	58	56
3	Surface	11295	11157	10616	11050		
	Grindability	68	67	58	61	47	51
4	Surface	8750	8748	8369	7502		
	Grindability	53	52	46	41	41	41
5	Surface	8845	7647	7371	5797		
	Grindability	53	46	40	32	31	31

[a] Averages of tests conducted by originating laboratory.

SHORTENING OF BALL-MILL PROCEDURE

About four hours are required for completion of a single grindability test by the ball-mill method, but by the use of a mechanical sieving device, and by proper coordination of milling and sieving, two samples can be completed in five hours. A shorter testing period would be advantageous, especially for control work. The most feasible way of shortening the procedure would be to grind a smaller proportion of the sample, if this could be accomplished without sacrificing the inherent accuracy of the method. To investigate this possibility 30 coals covering a wide range of grindability were selected, and their grindabilities were compared at each 10 per cent increment of finished product passing 200 mesh. The results at 10 and 20 per cent were erratic compared with the values at 80 per cent, particularly on the softer coals. At 30 per cent the agreement was better, and when 40 per cent of the sample had been ground to pass 200 mesh the agreement with the 80 per cent values was better still.

Table 13 shows determined and calculated grindabilities for 20 of the 30 coals considered. To avoid establishing a second grindability scale and to facilitate comparisons, the number of revolutions required to give 80 per cent through 200 mesh was calculated from the 40 per cent value by assuming a straight-line relationship between revolutions and percentages of through-200-mesh material. For the majority of coals the determined and calculated grindabilities agree very well. However, the calculated grindabilities of the first four coals show considerable error, due to the relatively greater amount of selectivity possible with these easily ground coals. For such coals the test should be carried to the 80 per cent end point to obtain reliable grindability values.

TABLE 13.—*Effect of Shortening Ball-mill Method*

Coal	Revolutions			Relative Grindability at 80 Per Cent through 200 Mesh	
	Determined		Calculated	Determined	Calculated
	40 Per Cent	80 Per Cent	80 Per Cent		
No. 1	281	596	562	100	100
Cranberry, W. Va.	273	596	546	100	103
Wilkeson 7, Wash.	272	612	544	97	103
No. 100	281	630	562	95	100
Adrian, Pa.	369	790	738	76	76
Wilkeson 4, Wash.	463	997	926	60	61
Unknown	489	1050	978	57	57
No. 2	521	1069	1042	56	54
Wellington, B.C.	504	1082	1008	55	56
No. 3	573	1163	1146	51	49
Bayne, Wash.	574	1217	1148	49	49
St. Charles, Mich.	606	1270	1212	47	46
No. 4	663	1342	1326	44	42
Utah	751	1513	1502	39	37
Bonny Blue, Va.	744	1540	1488	39	38
Low Splint, Ky.	758	1590	1516	37	37
Bellingham, Wash.	842	1700	1682	35	33
No. 5	977	1940	1954	31	29
Pa. anthracite	995	1995	1990	30	28
Centralia, Wash.	1009	2039	2018	29	28

When the grinding characteristics of a coal are well known, it should be permissible for control work to determine the number of revolutions required to grind 40 per cent through 200 mesh and calculate the 80 per cent value therefrom. The accuracy of the calculated grindability would be enhanced by application of a correction factor determined from the operator's experience with the coal in question. Calculated grindabilities should be checked frequently against determined values, by the full-length procedure.

SUMMARY

This report presents the results of further investigation of laboratory methods of estimating the grindability of coal. It compares the C.I.T. roll test, the Bureau of Mines ball-mill method, and the Hardgrove-machine method. The C.I.T. and Hardgrove methods specify a constant amount of work on each sample and attempt to estimate relative grindability from an approximation of the new surface produced by grinding only a small proportion of the sample to a fine size. The ball-mill method is a direct procedure involving measurement of the relative amount of energy required to grind coal to the size actually burned as powdered fuel.

The investigation showed that coals are mixtures of components, each of which requires a different amount of energy to grind. Relative grindability results cannot be accurate unless equal proportions of these components are ground to the size suitable for use as powdered fuel. This is accomplished by the ball-mill procedure, but not by either of the constant-work methods. With the latter type only the less resistant or softer components are ground; consequently the relative grindability values obtained do not represent accurately the grindability of the total sample.

ACKNOWLEDGMENTS

The authors are indebted to Alden H. Emery, assistant chief engineer, Experiment Stations Division, and W. H. Coghill, supervising engineer, Ore Dressing Section, Metallurgical Division, U. S. Bureau of Mines; to Milnor Roberts, dean, and Joseph Daniels, professor of mining and metallurgy, College of Mines, University of Washington; to Howard N. Eavenson, consulting mining engineer, Pittsburgh, Pa., and to J. B. Morrow, preparation manager, Pittsburgh Coal Co., for suggestions and helpful criticism.

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DISCUSSION

(*H. F. Hebley presiding*)

[This discussion refers also to the paper by C. G. Black, beginning on page 330]

W. A. Selvig,* Pittsburgh, Pa. (written discussion).—In the development of empirical laboratory methods such as the coal-grindability test it is necessary that the test procedure be standardized carefully so that different laboratories can duplicate results in check determinations on the same coal. It is equally important that the various factors that affect the test be investigated carefully if the test is to be of much value in testing grindability characteristics of coal in connection with the commercial pulverization of coal for use as powdered fuel.

Subcommittee VII on Pulverizing Characteristics of Coal of Committee D-5 on Coal and Coke, American Society for Testing Materials, selected for investigation four different grindability tests that were being used in the United States and Canada. Five coals, covering a wide range of grindability, from a soft semibituminous coal to a hard anthracite, were submitted to five laboratories for grindability determinations in duplicate. Each laboratory was given detailed instructions covering the procedure to be used for each of these methods. These cooperative tests showed that the Bureau of Mines ball-mill method and the Hardgrove method were the most precise; that is, these methods gave the closest agreement in duplicate determinations on the same coal by the same laboratory and by different laboratories. The tests indicated also that these two methods were the most accurate in determining grindability characteristics of different coals[11].

At a meeting of Subcommittee VII on Pulverizing Characteristics of Coal, held in New York on Feb. 21, 1935, the results of the cooperative grindability tests were given careful consideration and the subcommittee accepted as optional methods the Bureau of Mines ball-mill method and the Hardgrove method. At that time no data were available to determine definitely which of these two methods was the most accurate for estimating coal grindability. Both of these methods are now published as tentative standards by the American Society for Testing Materials with the view of giving them publicity and encouraging the use of them to determine which is the better method of test.

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[11] See W. A. Selvig: Check Determinations of Grindability of Coal by Various Methods. U. S. Bur. Mines *R. I.* 3301 (1936).

The authors of the paper have now presented data concerning the factors that affect these methods of test and which will be of help in determining the comparative merits of the two methods. The data concerning the effect of volume versus weight of the coal charge used as a sample is of particular importance in explaining what appeared to be abnormal results when testing the grindability of materials of widely different densities.

According to the authors, both of the methods use mills that grind selectively in regard to mixtures of hard and soft components of coal, but this difficulty is minimized in the ball-mill method by the large percentage of the charge that is ground to pass a 200-mesh sieve. Also, they are critical of the Hardgrove method because grindability is estimated from an approximation of the new surface produced by grinding only a small proportion of the sample to fine size, and that a relatively small percentage of material of subsieve size accounts for a relatively large amount of new surface produced. This suggests that closer agreement might be obtained between the two types of mills if the procedure of the Hardgrove method were changed to crush a larger amount of the sample, and the possibility of doing this by cycles with removal of the minus 200-mesh material as is done in the ball-mill method and calculating grindability according to the number of revolutions required to grind a definite percentage of the charge through a 200-mesh sieve. I should like to ask Dr. Yancey whether he has made any such tests using the Hardgrove grindability machine.

R. M. Hardgrove,* New York, N.Y. (written discussion).—While such theoretical studies as are reported in this paper are interesting, only the general use of both methods in practice will finally determine the most desirable and practical one to adopt as final standard.

Dealing with some of the comparisons in detail, the author has presented some determinations of subsieve sizes produced by the two methods and points out that the results obtained by the two methods agree somewhat more closely if correction is made for this variation. Again we wish to become practical and refer to the use to which grindability determinations are ultimately put; namely, the estimation of pulverizer capacities. The author apparently has overlooked two papers read before the A.S.M.E. in Chicago in 1933, one by Martin Frisch and the other by myself, dealing with the relation between grindability and pulverizer capacities. Fig. 16 of my paper shows the relation between grindability and capacity for a large number of different sizes of pulverizers. This indicates that capacity is proportional to grindability up to about 60 grindability but falls off at the higher grindabilities. This means that the grindability obtained by the Hardgrove method gives values too high for the softer coals to be proportional to pulverizer capacities. The ball-mill method gives results that are even higher at these higher grindabilities and are therefore still further away from being proportional to pulverizer capacities.

The data on using a constant volume confirms the use of the present methods using constant weight. For example, if in Table 10 the volume values are corrected for weight of material used the values practically check the values run on the weight basis.

The author states that the relationship between the ball-mill method and Hardgrove method results, as shown on Fig. 1, is impossible to establish. Ball-mill revolutions is a reciprocal unit and therefore should not be a straight line. If you actually plot the reciprocal curve where grindability is equal to 72,000 divided by the revolutions as instructed in the Tentative Standard for the ball-mill method, the test points fall quite close to the line except for the readings above 90 grindability. With the general use of grindability reported in per cent and thought of in that way by most people, it is very essential that results by the ball-mill method be reported in that way, and we would suggest that if the author does not like the simple constant now used in the Tentative Standard he present a more accurate mathematical formula that will fit the curve.

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* Babcock & Wilcox Co.

Actually the two methods are not so different as one might believe and modifications can be introduced into both. The machines used in the two methods are quite different in action and appearance but the biggest difference lies in the simplicity and ease of operating and cleaning the Hardgrove machine.

The 50-gram sample required by the Hardgrove method is much simpler to prepare than the 500-gram sample used in the ball-mill, and the smaller sample can be readily screened in a standard 8-in. screen used in mechanical screening.

The main feature discussed in this paper is the question of selective grinding. The author has tested mixtures of the hardest and softest coals he could find and points out that for these mixtures the ball-mill method is more representative than the Hardgrove method. While we must admit that coal is not a homogeneous substance, all the comparative tests made on individual coal samples have checked consistently, and this means that in actual practice the selective feature is not serious enough to affect the results. The actual cases where mixtures of very hard and soft coals are to be tested will be very rare.

There are materials other than coal that are decidedly nonhomogeneous, which do require more extended grinding to obtain representative results, and some lengthening of the method may ultimately be desirable, but on coal it is not necessary.

Perhaps we could briefly sum up the situation as follows: The Hardgrove method was developed as a direct laboratory tool by means of which results could be obtained at a reasonable cost, as an entire test can be made and calculated in less than one hour. The ball-mill method was later developed without regard for cost and time consumed and it requires four hours to run a test. Both methods give equal accuracy and consistency on coals as evinced by the comparative laboratory tests carried on last year. The Hardgrove method performs only a portion of the pulverizing act while the ball-mill method overgrinds or does more than is actually done in commercial practice. Dr. Yancey suggests that this method might be shortened to half the present number of stages and still obtain consistent results. This suggestion seems logical, since the Hardgrove method produces equally consistent results with a small amount of grinding. However, by shortening the ball-mill method, the error due to selective grinding is increased in proportion, a result that was not mentioned in the paper.

Compromises are inevitable and if the Hardgrove method were lengthened to cover several stages and the ball-mill method shortened to several stages, thereby permitting a test to be made at a more reasonable cost, the two methods would be on a more comparable basis. This compromise therefore strikes the most desirable balance between cost of making the tests and possible errors due to materials not being homogeneous.

The application of both methods to very fine grinding is a much more important problem than those covered in the present paper.

H. L. Brunjes,* New York, N.Y. (written discussion).—Since the adoption by the American Society for Testing Materials of the Bureau of Mines method as one of two alternative tentative standards for estimating the grindability of coal, we have examined our own test data for the purpose of determining whether or not the method could be shortened by stopping the process at an earlier cycle. Our conclusion is that an index based on the degree of pulverization prior to the fourth cycle gives erratic results that are not in sufficiently close agreement with the determined index based on carrying the process to the point where 80 per cent of the sample passes the 200-mesh screen. By stopping the process at the fourth cycle, however, we find it is possible to derive by calculation an index closely in line with that which results from carrying the test through to the end.

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The relationship between the number of revolutions at the 40 per cent point and at the 80 per cent point is not a direct proportion but is more accurately expressed by the formula $x = 2.055y + 47$.

TABLE 14.—*Grindability Tests by Fuel Engineering Company*

Sample No.	Number of Revolutions		Index Determined Based on 500 Revolutions = 100	Index Calculated from 40 Per Cent	Deviation of Calculated Index from Determined Index	Identity of Samples Used in Test		
	At 40 Per Cent	At 80 Per Cent				State	County	Seam
1	220	500	100	100		Pa.	Cambria	Lower Kittanning
2	225	505	99	99		Pa.	Cambria	Lower Kittanning
3	225	505	99	99		Pa.	Cambria	Lower Kittanning
4	230	530	94	95	+1	Pa.	Cambria	Lower Kittanning
5	235	520	96	93	−3	Pa.	Cambria	Lower Kittanning
6	230	535	93	95	+2	Pa.	Cambria	Lower Kittanning
7	245	560	89	90	+1	Pa.	Somerset	Upper Kittanning
8	265	575	87	84	−3	Pa.	Cambria	Upper Freeport
9	260	560	89	86	−3	Pa.	Cambria	Lower Freeport
10	260	585	86	86		Pa.	Somerset	Upper Kittanning
11	273	595	84	81	−3	Pa.	Cambria	Lower Freeport
12	280	605	83	80	−3	Pa.	Cambria	Upper Freeport
13	270	600	83	83		Pa.	Cambria	Lower Freeport
14	273	600	83	82	−1	Pa.	Cambria	Lower Freeport
15	284	625	80	79	−1	Pa.	Fayette	Lower Kittanning
16	285	630	79	79		W. Va.	McDowell	Pocahontas No. 3
17	283	630	79	79		W. Va.	Tucker	Upper Freeport
18	295	650	77	76	−1	Pa.	Cambria	Upper Freeport
19	285	655	76	79	+3	Pa.	Jefferson	Lower Kittanning
20	316	690	73	72	−1	Pa.	Indiana	Upper Freeport
21	314	680	74	72	−2	Pa.	Jefferson	Lower Kittanning
22	300	690	73	75	+2	Pa.	Jefferson	Lower Freeport
23	316	710	70	72	+2	Pa.	Jefferson	Lower Freeport
24	330	720	70	69	−1	Pa.	Westmoreland	Pittsburgh
25	345	740	68	66	−2	Pa.	Clearfield	Lower Freeport
26	350	770	65	65		Pa.	Fayette	Pittsburgh
27	380	825	61	60	−1	W. Va.	Randolph	Kittanning
28	390	850	59	59		Pa.	Jefferson	Lower Freeport
29	380	870	58	60	+2	Pa.	Jefferson	Lower Freeport
30	405	870	58	57	−1	W. Va.	Webster	Sewell
31	410	890	56	56		W. Va.	Monongalia	Pittsburgh
32	406	905	55	57	+2	Pa.	Jefferson	Lower Kittanning
33	410	910	55	56	+1	W. Va.	Randolph	Sewell
34	433	925	54	53	−1	Pa.	Westmoreland	Pittsburgh
35	427	950	53	54	+1	Pa.	Westmoreland	Pittsburgh
36	425	930	54	54		W. Va.	Webster	Sewell-Cannel
37	425	955	52	54	+2	Pa.	Jefferson	Lower Kittanning
38	445	960	52	52		Pa.	Allegheny	Twin Freeport
39	465	990	51	50	−1	W. Va.	Marion	Pittsburgh
40	440	985	51	52	+1	Pa.	Fayette	Pittsburgh
41	447	980	51	52	+1	Pa.	Westmoreland	Pittsburgh
42	471	990	51	49	−2	Pa.	Allegheny	Pittsburgh
43	449	985	51	51		Pa.	Elk	Lower Kittanning
44	464	985	51	50	−1	Pa.	Allegheny	Pittsburgh
45	461	1000	50	50		W. Va.	Monongalia	Pittsburgh
46	478	1020	49	48	−1	Pa.	Washington	Pittsburgh
47	479	1030	49	48	−1	W. Va.	Brooke	Pittsburgh
48	476	1040	48	49	+1	Pa.	Westmoreland	Pittsburgh
49	481	1045	48	48		W. Va.	Marion	Pittsburgh
50	469	1020	49	49		W. Va.	Monongalia	Sewickley
51	493	1050	48	47	−1	W. Va.	Monongalia	Pittsburgh
52	508	1060	47	46	−1	Pa.	Washington	Pittsburgh
53	540	1140	44	43	−1	Ill.	Franklin	No. 6
54	526	1160	43	44	+1	Tex.		Lignite
55	570	1230	41	41		Ill.	St. Clair	No. 6
56	584	1260	40	40		Ky.	Floyd	Elkhorn
57	612	1310	38	38		Pa.	Washington	Pittsburgh
58	626	1345	37	38	+1	Pa.	Northumberland	Anthracite
59	635	1360	37	37		Ky.	Floyd	Elkhorn
60	641	1370	37	37		Ky.	Harlan	High Splint
61	702	1490	34	34		Ky.	Harlan	High Splint

In this formula y equals the number of revolutions required to reduce 40 per cent of the sample to minus 200 mesh, a figure that is determined by interpolation between the third and the fourth cycle. Curves of the grinding rate show that the reduction in the size of the particles is more nearly a direct proportion to the number of revolutions between the second and the fifth cycle. After that point the curve slopes off slightly from a straight line at a slightly different rate for different coals. It is the difference in this rate of grinding in the last two or three cycles that occasions the deviation between the calculated and the determined index. Whether the difference in degree of deviation from a straight line in the later cycles is due to a mill characteristic or to differences in the coal is not clear to us. It seems, however, that possibly a change to a constant-volume basis in place of the present constant weight of samples might modify this in the direction of greater accuracy. We had hoped to make some experiments along that line in time for this discussion but we have been unable to do so.

In Table 14 we give the results of 61 grindability tests made in our laboratory, showing the number of revolutions at the 40 per cent point, at the 80 per cent point and the index of grindability derived by applying this formula as compared with the index determined from the completed test. In making these comparisons we have found it more satisfactory to adopt a definite amount of work (for example, 500 revolutions) as the reference point and considering that coal that requires 500 revolutions for the completed test, to have an index of 100. By doing this instead of using the revolutions at either the 40 per cent point or the 80 per cent point for a particular coal that happens to be the easiest to grind in a particular series, it is possible to derive a formula that is applicable to the work of other investigators, irrespective of the particular coals

TABLE 15.—*Data of Table* 13 *Recalculated*

From Table 13		Calculated 80 Per Cent, Fuel Engineering Company Formula	Deviation of F. E. Co. Formula from Determined Index
Sample No.	Determined 80 Per Cent		
1	84	80	−4
2	84	82	−2
3	82	82	
4	79	80	+1
5	63	62	−1
6	50	50	
7	48	48	
8	47	45	−2
9	46	46	
10	43	41	−2
11	41	41	
12	39	39	
13	37	35	−2
14	33	31	−2
15	32	32	
16	31	31	
17	29	28	−1
18	26	24	−2
19	25	24	−1
20	25	24	−1

being tested by the several investigators. Table 14 shows that the calculated index in 16 cases is above the determined index with an average deviation of 1.5; 23 have a calculated index below the determined index, with an average deviation of 1.6 and 22 have the same index by either method. The average deviation for the entire series is ∓ 0.98.

Table 15 shows the data contained in Yancey's Table 13 recalculated, using 500 revolutions = 100, and applying our formula to his 40 per cent point. Considering the fact that we are here applying a formula derived from an entirely independent series of tests, for the most part on entirely different samples, the agreement is very satisfactory. Eight of the 20 samples show the same index from the use of the formula as from the determined result; five are ∓ 1, six are ∓ 2 and only one, the first sample in the series, shows a substantial deviation of minus 4.

It is noticeable, however, that both in the Yancey series of tests and our own, the maximum deviations are confined to the coals in the higher range of grindability where such deviations are of the least practical significance.

We believe that further work along the line of closer refinement of the formula suggested here with improvement in testing methods possibly including a change to a sample of constant volume should be carried out, because there is real promise of shortening this method, without any significant loss in accuracy.

F. A. JORDAN,* Youngstown, Ohio.—Is this index finally going to be applicable to all materials that we grind in the mineral industry? I can see that if it is, it would very nicely help in determining cost of grinding by comparing grindability of some materials, and in that way arrive as to relative differences in the grinding costs.

R. M. HARDGROVE.—We have applied the method to various other materials rather extensively. I believe we have in our files at the present time tests of over 200 cement clinkers and an equal number of cement rocks, obtained from various parts of the country. We have not gone into ores as much as we might have, but have tested a number of other materials such as talc and phosphate rock. Thus we have accumulated a lot of information and it is of great help in applying pulverizers to those miscellaneous materials.

C. G. BLACK.—In regard to Mr. Hardgrove's discussion on hardness and the cost of grindability, I might say that in our work on grindability we also ran samples of pyrite and marcasite, which are very hard, and we used a 500-gram weight of those materials and found them to require around 1200 revolutions, or similar to a good many coals. It would seem to me that a test that shows pyrite and marcasite just as easy to grind as coal in an air-swept mill, that is, in pounds per hour, or tons per hour, does not give the true picture; in other words, I should hardly expect that marcasite or pyrite could be ground for the same cost as some of our coals, and this would also apply to Mr. Hardgrove in the use of his machine on materials outside of coal.

It is interesting also to know of some of the work done by Dr. Heywood in England. One of his tests was called an attrition test, which is similar to the Hardgrove, and in his work he also mentions the use of volume instead of a weight measure in dealing with denser materials compared to coal.

R. M. HARDGROVE.—We have tested pyrites and find the grindability runs 40 to 45 on our usual scale. In pulverizing other materials, such as cement clinker and cement rocks, which we know are harder than coal, we find, for instance, that cement rock, having a specific gravity of approximately 2.75 and a grindability on our scale of 70, will give the same capacity on a pulverizer as coal of 1.30 sp. gr. having a grindability of 70, so that I think it is true that you do get the same capacities when pulverizing these very hard, dense materials or soft lighter materials, if the grindabilities are the same.

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* Youngstown Sheet & Tube Co.

F. A. Jordan.—I have had much experience in grinding hard material and I have the impression we can grind a ton of hard material for less money than we can grind a ton of coal. Mr. Black said 1200 for pyrite. I think he can grind a ton of pyrite for less money than a ton of Banning coal, and I raise the question whether the number of particles that are inversely as the specific gravity does not have a very considerable part in the grinding.

C. E. Lesher,* Pittsburgh, Pa.—The following question has been raised: Why are we interested in these things? Perhaps from the standpoint of the manufacturer of equipment for grinding there is a point in having a comparison between the grindability of cement clinker, of pyrite, and of coal, but I have a suspicion that the intense interest in the grindability of coals has come about for another reason.

Some sections of the coal industry have assumed that it is perfectly feasible to correlate all coals with respect to the value at point of consumption. The idea began when there was price fixing under the NRA.

When the commercial men in the coal industry were asked whether or not coals could be fairly and properly priced, they ran into some rather difficult problems and they turned those problems over to the technician, who is represented here.

It was proposed, for instance, that differences in value of coals arose with differences of grindability and the attempt was made in 1933 and 1934 to correlate coals on these properties. Of course, these methods of correlation were carried to absurd degrees of refinement. Gold miners can determine the fineness of their bullion with extreme accuracy, but when the coal producer talks of the grindability of his coal in similar terms and says it is an important factor in his price, then indeed he is in trouble. I think that explains why we had so much technical research in the establishment of methods.

To come back to my original point: We are interested in these things because there are yet those who believe it is possible and practical and proper to establish the value of the coal price by mathematical formula instead of by the time-worn method of letting the customer decide as to the coal; that is, between respective coals in his particular plant; his own usability.

H. Heywood,† London, England (written discussion).—The paper by Dr. Yancey and Mr. Geer is of particular interest to the writer, who constructed a ball mill to the Tentative Specification of Committee D-5 of the American Society for Testing Materials as soon as this was issued. The results of tests made on a wide range of coals were very satisfactory, and were compared with other tests in a paper presented by the writer to the Institute of Fuel in December, 1935 (vol. 9, p. 94).

Experiments by the writer confirm that the charge to the mill should be of constant volume, otherwise the results on materials that are denser than coals are entirely misleading. The test results shown in Table 16 illustrate this point. These tests were made in the standard manner by grinding in 10 per cent cycles, and show that if 500 grams of both coal and shale are taken as the charge, these materials would appear to have approximately equal grindability; whereas actually the shale is nearly twice as difficult to grind as the coal.

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* Pittsburgh Coal Co.

† Mechanical Engineering Dept., The Polytechnic.

TABLE 16.—*Tests on Soft and Hard Materials*

1 Material	2 Specific Gravity	3 Weight of Charge, Grams	4 Revs. for 80 Per Cent through 200 Mesh	5 Weight of Charge, Grams	6 Revs. for 80 Per Cent through 200 Mesh	7 Ratio of Columns 6:4
Barnsley soft coal........	1.27	250	575	500	1020	1.77
Shale....................	2.47	500	1050	1000	1980	1.88
Pyrites..................	4.56	875	1760	1750	3460	1.96

The other point of extreme interest in the paper is the selective action when grinding a mixture of materials. Some early experiments made by the writer on mixtures of coal and shale ground in a laboratory preparation mill may be of interest in this connection. Three mixtures were made, containing respectively 10, 30, and 50 per cent by weight of shale. The samples were ground in cycles of 20 revolutions of the mill, and after each cycle were sieved on a 100-mesh sieve; the residue was returned to the mill together with sufficient new feed to counterbalance the weight of product passing 100-mesh sieve. The method thus corresponds to the equilibrium method of Maxon, Cadena and Bond. After seven cycles, the percentage of shale in the residue and in the product passing 100-mesh sieve was determined by float and sink separation. The results were as follows:

Shale in Feed, Per Cent	Shale in Residue, Per Cent	Shale in Product, Per Cent
10	13.0	4.5
30	37.7	20.0
50	60.6	30.2

It would thus appear that a satisfactory grindability test on a raw coal containing 10 to 20 per cent of shale can only be made by separating the two constituents by means of a liquid of suitable density, and making a ball-mill test on each constituent. The results of the two tests can then be combined by calculation according to the proportion of each constituent present in the raw coal.

With reference to the Hardgrove test, the writer found that the surface of the fine material passing a 300-mesh sieve varied with the nature of the material. Although there was not a very great difference between coals, shale and pyrites differed considerably from coal, the through 300-mesh material being about three times as fine as for coal. It is not surprising, therefore, that results of Hardgrove and ball-mill tests do not agree on such materials. The difference is due to the Hardgrove method rather than to the machine, and if the Hardgrove test were made in cycles of grinding and sieving, the residue only being returned to the mill, the two tests should be in much closer agreement.

H. F. YANCEY AND M. R. GEER (written discussion).—Mr. Selvig inquires whether cycle tests of coal have been made with the Hardgrove machine. We have done a little work along this line, but none that is strictly comparable. Two coals, Nos. 1 and 5 (Table 4), were ground in the Hardgrove machine, using the alternate grinding and screening procedure of the ball-mill method, except that a 300-mesh sieve was used for removing undersize instead of a 200-mesh. The grindability value was taken as the number of revolutions required to grind to 80 per cent through 300 mesh. With this procedure, coal No. 5 was 18 per cent as grindable as coal No. 1 in contrast to the 31 per cent obtained by a regular ball-mill test.

Mr. Hardgrove refers to the relative value of theoretical studies; nevertheless this method of approach is the only one available for determining the accuracy and fundamental principles on which the proposed methods are based. General use of a method can demonstrate only its ease of operation, time required, and incidentally its reproducibility. An inaccurate method, even though reproducible and rapid, should not be preferred to a longer procedure that is both accurate and reproducible.

We do not regard the comparative tests listed in Table 4 as consistent checks, although they are so described by Mr. Hardgrove. The difference of 11 per cent between the ball-mill and Hardgrove values for coal 2, and the smaller but still significant difference for coals 3, 4 and 5, are much too large to neglect.

There seems to be considerable confusion regarding the presentation of ball-mill test results and correlation of ball-mill and Hardgrove grindability values. We agree with Mr. Hardgrove that it is desirable to express ball-mill results in percentage. Any percentage scale must have as its base an actual or hypothetical standard coal to which all others are referred. The Hardgrove scale is based on a coal which gives 656 new surface units in a Hardgrove test. The same coal, tested by the ball-mill method, requires 630 revolutions to reach the end point specified in that method. If the factor 72,000 is used to convert ball-mill revolutions to percentage grindability, all coals are compared to a hypothetical standard requiring 720 revolutions. It is obvious that results by the two methods are not directly comparable when one method uses a standard of 630 revolutions and the other a standard of 720. Direct comparison is made possible only when a common standard is used. In this paper coal No. 1 was adopted as a temporary standard for many of the comparisons; Hardgrove's standard might have been used by dividing ball-mill revolutions into the factor 63,000. Mr. Brunjes has suggested the use of the factor 50,000 to convert ball-mill revolutions to percentage grindability because it has the advantage of simplicity and it places all coals under 100 per cent.

The paper Mr. Hardgrove read before the American Society of Mechanical Engineers in 1933 has been carefully studied. When calculated on the proper basis, as just explained, the ball-mill percentage grindabilities for softer coals are lower rather than higher than the corresponding Hardgrove values and consequently, when substituted in Fig. 16 of his paper, place the capacity curve in better alignment with the grindability curve than do the Hardgrove values. Mr. Hardgrove has thus provided an excellent example of how misleading the factor 72,000 may be in comparing the two methods. It illustrates the point that direct comparisons can be made only when a common standard is used for the two methods.

Dr. Heywood's contribution to this discussion is particularly gratifying, not only because of his previous fundamental investigations in the field of fine grinding but because his supporting data confirm our principal conclusions. The last column of his first table (Table 16) shows that the efficiency of the mill varies with the volume of mill charge. If the efficiency were constant each of the values would be 2.00. Dr. Heywood suggests a separate test of the shale or other heavy impurity in a dirty, raw coal. Although this procedure is proper it is probably too time-consuming for a routine test, in which the only feasible way of controlling selectivity lies in grinding a large proportion of the sample to finished size.

The data included in Mr. Brunjes' discussion are a valuable contribution to the paper. The coals tested cover a wide range in grindability and are sufficient in number to provide a sound basis for shortening the method. His tests, together with those of C. G. Black[11] and our own, demonstrate that the method can be shortened without significantly impairing its accuracy. The formula suggested for calculating the number of revolutions at 80 per cent through 200 mesh from the value at 40 per cent can be made applicable to any group of coals by proper selection of the factors, and for that reason has been included in the proposed revision of the tentative A.S.T.M. method.

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[11] See page 330, this volume.

As Mr. Hardgrove says, shortening the method makes it more subject to selectivity. This feature is taken care of by the correction factor. It should be borne in mind also that a full-length test is advocated for the softer coals when a high degree of accuracy is desired.

To clarify some of the points that have been raised regarding the use of equal volumes of mill charges, it is necessary to consider the fundamentals on which both methods are based. The Rittinger law states that in grinding a given material the amount of net energy required is proportional to the new surface produced. It follows that the net energies required to produce equal surface areas in different materials are proportional to their grindabilities, or that for a given expenditure of net energy the surface areas produced in different materials are proportional to their grindabilities. Measurement of the net energy used in grinding is difficult with small laboratory mills; hence both methods assume grindability to be proportional to the gross energy supplied to the mill, that is, to the number of revolutions of the mill. The validity of this assumption depends on maintaining a constant ratio between gross and net energy in different tests; that is, constant grinding efficiency.

TABLE 17.—*Effect of Variation of Volume of Mill Charge on Efficiency of Grinding a Given Coal*

Sample Weight[a], Grams	Ball-mill Method				Hardgrove Method			
	Revolutions	Through 200-mesh, Grams	Grams per Revolution	Percentage of 500-gram Rate	Revolutions	Through 200-mesh, Grams	Grams per Revolution	Percentage of 50-gram Rate
135	450	108	0.240	70.6	60	3.45	0.0575	37.3
255	658	204	0.310	91.2	60	5.95	0.0992	64.4
465	1067	372	0.349	102.6	60	6.63	0.0111	72.1
500	1175	400	0.340	100.0	60	9.24	0.1540	100.0
1060	2687	848	0.316	92.9	60	12.00	0.2000	129.9

[a] Sample weights used in Hardgrove tests were one-tenth of those shown in this column.

Tables 16 and 17 demonstrate that in either type of mill the grinding efficiency varies with change in volume of mill charge. It follows that to insure accurate estimation of the net energy used in grinding, a constant volume of mill charge must be used. With a mill grinding various volumes of the same coal, efficiency becomes the ratio of gross energy to new surface and may be approximated by the weights of through 200-mesh material produced per revolution of the mill. Table 17 shows that in both the Hardgrove machine and the ball-mill tests of various weights of the same coal, grinding rate, and hence efficiency, varies with change in volume of mill charge.

A second phase of the equal volume discussion is complicated by two divergent conceptions of grindability. One may consider grindability as a property which in reality is a composite of such properties as hardness, toughness, and fracture characteristics. This is the grindability to which, in Rittinger's theory, the energies required to produce equal surface areas are proportional, and which for this discussion might be termed "absolute" grindability. Such a grindability value may be used to characterize a coal in much the same way as the value 7 describes the hardness of quartz on Mohs' scale. A second and different concept of grindability is that it is a value from which mill capacity can be estimated. Because mills are rated in tons of material of a specified fineness per horsepower-hour rather than in square yards of new surface area per unit of work, this second concept regards grindability as the quantity to which are proportional the energies required to produce equal weights of materials of the same fineness. This might be termed "specific" grindability. Absolute grindability, then, is the number of revolutions, or the energy required to grind equal volumes of materials to the same fineness, and specific grindability must be calculated from this value by taking the density of the material into consideration.

These two concepts of grindability, absolute and specific, have been described to explain the problem involved in choosing the type of value that should be used to express the results of grindability tests. Specific grindability values are of use only to those interested in the capacity of a mill grinding various materials, whereas absolute grindability is the property thought of and recognized by most persons. These two concepts of grindability are illustrated in the statements of two men. A coal operator writes that any grindability method that shows the impurities in coal to be easier to grind than coal is misrepresenting the facts. A mill manufacturer's engineer states that a mill will have the same capacity whether grinding coal or rock, provided that their grindabilities are the same. The coal operator is thinking of absolute grindability, a property similar to hardness, whereas the engineer is considering specific grindability. Whether a given impurity has a higher or lower grindability than coal depends partly on the definition of grindability. The criticism of this paper indicates that the most general use of grindability determinations is in estimation of mill capacities.

Correlation of commercial mill capacities with grindability values is not within the scope of this paper, but it should be pointed out that the estimation of capacity depends on so many variables other than grindability that the utmost discretion should be practiced in using the test values for this purpose.

Two conclusions may be stated: (1) With materials of varying density equal volumes of mill charges must be used in both methods to insure equal grinding efficiencies on each material tested; (2) correlation with mill capacity is facilitated if the results obtained on equal volumes are converted to a weight basis. With the ball-mill method this may be done by multiplying the number of revolutions required to grind 80 per cent of the standard volume through 200 mesh by the ratio of 500 to the weight in grams of the standard volume. The change in mill efficiency occasioned by the relatively small variation of specific gravity between most coals used as pulverized fuel has little effect on grindability determinations made on an equal weight basis. For this reason use of equal volume charges in testing coals seems unnecessary, but equal volumes must be used in testing materials with a specific gravity very different from that of coal.

Regulation of the Coal Industry

By Howard N. Eavenson,* Member A.I.M.E.

(New York Meeting, February, 1936)

One who has been trained in belief in the law of supply and demand and its effect upon prices finds it difficult to adjust himself to the minute regulations imposed by the New Deal, and also to the bewildering changes of government policy as expressed within a short period of time between the antitrust laws and the N.R.A., and later by the Bituminous Coal Conservation Act of 1935. A survey of the coal industry since the World War, both in this country and Great Britain, however, shows that regulation of some sort is needed if any stabilization is to be achieved and if the wage structure and economic welfare of the many communities dependent solely upon coal are to be maintained.

Let us look at what has happened to the industry since 1920, which can be considered as the end of the period affected by the war influence. This discussion will be confined solely to bituminous coal, although many of the general conditions are equally applicable to anthracite, the position of which now is at least as bad as that of the softer variety.

Since 1920—to the end of 1934—the production of coal has decreased from 568.7 million to 358.4 million tons, or 37 per cent; the number of mines from 14,766 to 6,258, or 58 per cent; the average annual production per mine has changed from 38,512 to 57,425 tons, or 50 per cent. The number of large mines has increased considerably, and the number of men employed has decreased from 639,547 to 458,011, a reduction of 28 per cent, while the average output per employee has changed from 4.0 to 4.4 tons, or an increase of 10 per cent. The average price per ton of coal at the mine has varied from \$3.75 to \$1.75. During that time wages decreased largely until 1933 and since that time have increased, until now the rates paid are considerably above those in other occupations, although the actual yearly earnings are low, because of part-time work. From 1920 to the signing of the Jacksonville wage agreement in 1924, about two-thirds of the men were union members and practically all of the nonunion mines were south of the Ohio River; with the fixing of the high wage scale of that agreement, which soon proved to be an uneconomic one, many of the mines found that they could not retain their business, so gradually changed to nonunion. By 1928 the proportion

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of union men had dropped to about one-third, where it remained until the N.R.A. law was passed in June, 1933, since which time at least 90 per cent of all coal miners have belonged to the United Mine Workers of America.

Appalachian Coals, Inc.

In the 11 years 1923 to 1933, the position of the industry grew steadily worse; all through the boom years of the twenties and in 1930 wages were so low and conditions generally so bad that the governors of many of the major coal-producing states urged the leaders of the industry to take steps to remedy them. After many conferences and much work it was found that the only plan that the industry thought legal and feasible at that time was the sales agency plan, under which the mines in any district would consolidate their sales efforts into one company in order to reduce competition and stabilize prices, with many other expected advantages. This plan was formulated in the fall of 1931, and in March, 1932, eight southern high-volatile districts, having 137 producers and a total production of 58 million tons, agreed to start operations under the name of Appalachian Coals, Inc. At the stockholders' organization meeting a telegram was received from the Department of Justice stating that the plan was illegal and that a suit would be brought to prevent its consummation. A long trial before a three-judge court in August, 1932, resulted in a decision sustaining the Department's contention, but an appeal to the U.S. Supreme Court resulted in March, 1933, in a permissive decree that the plan was legal if executed as stated in the evidence, and if competition was maintained within the districts represented in the agency, and with other districts. The organization was perfected and prices announced in April, and by June conditions had so improved that a wage increase was made, followed by another two months later.

It was early found that companies not included began "chiseling" on prices and that these could not be maintained as expected, on this account. It was intended in the original contract that working time would be equalized, if possible, between mines and districts. Many things prevented the putting into effect of this provision, among them being the N.R.A. act in Congress, the unionization of the mines, formation of the code and wage negotiations; it proved impossible to secure promptly the organization desired, and the actual operation of the Code Authority, which included all mines in Appalachian Coals, Inc., as well as those outside, further prevented it. Much dissatisfaction was created by the lack of equalization of running time, but no workable plan was devised to do it, and most operators thought it impossible of accomplishment. The efforts to do it by lowering prices for certain mines had little effect on the situation.

While under the N.R.A. Appalachian Coals, Inc. was the price-fixing agent for that Code Authority, and acted in this capacity during the life of the Code, there were a greater number of mines outside of A.C.I. than in it that were directly under the Code Authority, although their average output was much less than that of the Appalachian members. These mines were handled directly by the Code Authority, which resulted in duplication of a considerable amount of work, and by the time the Supreme Court decided the N.R.A. unconstitutional, the dissatisfaction among A.C.I. members had materially increased, and for a number of reasons it has remained inactive since that time as far as price fixing is concerned, although a great deal of valuable work has been done along the lines of trade promotion and engineering work on combustion.

Most of the members of Appalachian feel that the idea never had a thorough trial, because of the interference by N.R.A. operations. Cooperation between the two bodies was rendered difficult because A.C.I. was operating under a permissive decree which prohibited it from cooperating with other districts, as was required under the Code. On account of the various causes mentioned, and particularly of the fact that some of the large producers felt that the operation of A.C.I. had resulted in a decrease of their output and that they were not as well off, even under the regulated prices, as they would otherwise have been, it is doubtful whether under the conditions now existing Appalachian Coals can be revived as originally intended, if the so-called Guffey Bill is declared unconstitutional. Final determination of this depends, of course, entirely on the condition of the coal market when the situation develops, which probably will not be until the middle of this year.

Bituminous Coal Conservation Act

Under the Code the great majority of coal operators were materially benefited, but the Code broke down in its price fixing because of lack of enforcement of the prices, the difficulty of coordinating those fixed in one district with those fixed in another; and because no central, unprejudiced body was provided to review decisions and secure cooperation between districts. During the period immediately following the breakdown of the Code in the early part of 1935, and until the present time, prices of coal, however, have again reached a low point, and on many qualities of coal they are as low now, considering the increased cost due to higher wages, as they were in 1932.

Approximately 70 per cent of the coal production of the country has accepted the Code under the Bituminous Coal Conservation Act of 1935, the remaining 30 per cent consisting of nearly all the captive companies operated by the steel companies, which is by far the largest proportion of captive tonnage, and most of the largest producing companies, which never believed in the Guffey Act as passed and have been vigorously fighting it since it became effective. Captive companies rarely sell any coal outside, so that ordinarily their tonnage does not compete with that of the commercial companies, although there is always the possibility that they will so sell their coal. In the opinion of most coal operators, it is going to be extremely difficult for the price-fixing provisions of the Act to be made effective with many of the largest and most aggressive companies in the country fighting it vigorously. This opinion may be wrong, and only time will tell the outcome.

The Bituminous Coal Conservation Act of 1935 should have been called a stabilization act, as, excepting the statement in the preamble of the bill about the conservation of coal, there is absolutely nothing in the law that will conserve bituminous coal resources, although if it becomes profitable to remove coal that is now being left in the mine this may be done. There is nothing in the law to require it, however. Unquestionably many mining companies are now extracting only the cream of their coal, because it is economical to do so, and, unless prices advance sufficiently to warrant the increased cost of production, this practice will be maintained. With practically no exceptions, no pillar coal is recovered in the entire state of Illinois, and this practice is gradually increasing in southern West Virginia and Kentucky. In many mines the cost of complete extraction is materially greater than that when the pillars are not removed, so that the loss of coal, while harmful from a national standpoint, is economically beneficial to the operating companies. Under present decisions and the division of authority between the National and State Governments, there is absolutely nothing that can be put in an act of this kind that will require the removal of this coal.

Regulation in England

It is interesting to note that our English friends are passing through the same cycle of coal regulation that we are experiencing, except that in England the movement started some three or four years earlier than in this country, and it has operated with as little government supervision as possible, although the Act did enable the Government to make compulsory consolidations in districts, if necessary. This portion of the Act did not work satisfactorily, and the consolidations have been comparatively few in number. The main purpose of the Act in England was to enable export coal, aggregating perhaps 20 per cent of their output, to be sold at lower prices than domestic coal, and this portion of the Bill has operated fairly satisfactorily, although because of the difficulties of arranging quotas, etc., some dissatisfaction exists among operators. Lately the Government has insisted that the coal operators organize central sales agencies in each district, and all of the different fields have agreed to put such sales agencies into effect prior to July 1, 1936.

The requirements the Government demands in the new schedule for selling schemes are as follows:

1. They must cover all of the coal owners in a district and have a measure of permanency.
2. They must effectively prevent intercolliery competition; and
3. They must be so drawn that evasions cannot exist.

To anyone who is familiar with the operations of such plans in this country this seems to be a large order, but, of course, there are no anti-trust laws in Great Britain, and with the active support of the Government and the greater tendency to observe laws in that country, it may be possible to work up some scheme that will satisfactorily meet these conditions.

Problems in the United States

Here, opportunity for coal regulation is complicated by the presence of oil and gas, the combined amount being nearly equal to the solid fuel. Any regulation of coal, unless also applied to the liquid fuels, will inevitably greatly reduce the volume sold, especially as freight charges are a much greater proportion of the delivered price of coal than of oil and gas. Another factor is that the freight rate structure for coal is an extremely rigid one, while pipe lines are subject to no regulation, and for several years in the great cities of the Middle West we have seen natural gas, in great quantities, sold at much less than the cost of transportation. Unless any regulation of coal is equalized, either by similar treatment of other fuels or by taxes of equivalent amounts, the sales of coal in competitive territory will gradually dwindle.

Experience of the last few years has shown the industry conclusively that no price stabilization can exist without wage stabilization. The enormous number of small operations, many of them in isolated localities, furnishes the opportunity to reduce costs by depressing wages. Facing heavy losses, many operators seize this chance of maintaining tonnage by cutting prices first and then wages, and once started this cycle can lead only to conditions deplorable in every respect, and enjoyed least of all by the operators themselves. As long as the union keeps its present strength, maintains discipline among its members and does not force wages to uneconomic levels, the present situation greatly helps the industry. The Guffey bill was written to control this situation by law, and if it is declared unconstitutional—and according to precedent the wages and hours provisions will almost certainly be so called—the present union contract will maintain the status quo. Sooner or later, some regulation of the union power must be attempted, or eventually the bituminous industry will again reach the position it had in a large part of the country under the Jacksonville scale, and which the anthracite industry now has, of having an uneconomically high wage scale—and a greatly diminished business.

If the Guffey law is declared unconstitutional, as many of our lawyers feel it must be, it is evident that little can be done for regulation by the Federal Government by compulsion. What then can be done?

The Appalachian Coals decision shows how far the Supreme Court has gone to meet the conditions in the industry under the existing antitrust laws. To have helpful regulations, the following conditions must exist:

1. Districts must be able to unite, to fix prices and establish quotas if necessary.
2. No new mines should be opened, or old ones be reopened, until their production is needed.
3. As many operators as possible in each district should be included.
4. Wages and hours of work should be maintained at reasonable standards.
5. The public must be protected from both operators and labor.

The first condition can be met by repealing the antitrust laws so far as the natural-resources industries are concerned, encouraging the formation of such district agencies and allowing them to make agreements with each other for prices and output quotas, but all such agreements to be subject to approval of a Federal Commission to protect the public interest. This would allow the control of the business to be in the hands of the operators, and the public would be protected by the action of the Commission.

The second condition is the hardest to meet, under our laws, and the most essential to success. It is believed that Congress could instruct the Interstate Commerce Commission to refuse permission to the railroads to connect new sidings, or to renew old ones, except to maintain production of existing companies, for a limited period—say five years—while the present condition of overcapacity exists. This would destroy no property but merely defer its use for a limited period. This could be extended until production approached more nearly to capacity.

As long as human nature is what it is, the third condition is difficult of fulfillment. It will be possible only when operators look more closely at the profit than at the tonnage figures. When they become profit-minded on a smaller production, rather than tonnage-minded on no profit, it can be done. The writer will never see this time, but it must be said for the industry that more operators have this viewpoint now than ever before. In the meantime fair trade regulations, including one making it an offense to sell habitually below cost of production, would be helpful, particularly if they could be enforced by a body handling only natural-resources industries.

The question of wages and hours is settled for the present, and the public interest has been protected.

Such regulation would not repeal the law of supply and demand, but if properly administered would relieve many of the hardships of that age-old principle; it would tend to reduce costs of distribution—now a serious evil; it would help to consolidate the plants into more economically operated units and thus to reduce costs; it would stabilize industrial conditions in many communities where no other industries exist and make working and living conditions more in keeping than they have been with what thinking operators want, and would insure a constant supply of fuel of the kinds wanted at all times.

If all the fuel industries are regulated, or costs equalized by taxation, as suggested, it can be done with little change in our legal structure, and would be voluntary, instead of compulsory. It would not be perfect, and would not be satisfactory to all, but was there ever any law, anywhere, at any time, that was?

DISCUSSION

(*Arthur Notman presiding*)

W. O. Hotchkiss,* Troy, N.Y.—There is one thing in the paper on which I should like to hear Mr. Eavenson elaborate as to possibilities. That is the question of equalizing by taxation. That can only be done by state law probably, and in most cases the interests of states, as I can see it, are averse to any such thing. The large oil-producing states do not produce coal and probably would object to any increase in taxation that would hinder their sales. Is there any feasible way for that to be done that you know of?

H. N. Eavenson.—What I had in mind, and what I might say is also being talked of among leaders of the industry—in fact, the bill may be presented to this Congress—is asking Congress to put a tax on fuel oil, of something like two or three cents a gallon, or whatever it might be, and also a tax on natural gas that is transported in interstate commerce at approximately five cents per M.cu.ft. That would take care of the increase in the cost of coal caused by Federal regulation, and is something that was promised when NRA was passed.

At that time the injustice of increasing the price of coal without similar increases in its competitors was pointed out. Nothing was ever done about it except to increase the cost of production of coal. I think Congress undoubtedly has the power to levy such taxes. Oil coming into the country, gasoline and lubricating oil are already taxed. It is just a question of adding fuel-oil tax similar to taxing lubricating oil.

There is no question about the right of Congress to regulate the movement of natural gas in interstate commerce. That is all the coal industry is interested in, and it does not care about local gas consumption. We now have enormous pipe lines that are dumping natural gas at less than the cost of transportation into the great markets that the coal business has always had. If the delivered cost of natural gas were the actual cost of transportation plus the cost at the wells, there would be little trouble about the competition. Natural gas is delivered in Chicago and sold in great quantities for 15 or 18¢, and sometimes as low as 12¢ a thousand feet. In Minneapolis and

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* President, Rensselaer Polytechnic Institute.

St. Paul the sales price is 18¢, but in all of that central territory coal cannot compete with such prices. A line is to be built into Detroit, which is a very large coal market, but whether such low prices will be possible there, I do not know.

That is the thing that was in mind, the equalization of taxes. Not only is it needed but it can be done if Congress will pass the law.

A. W. Lawson, Chicago, Ill.—Do I understand, Mr. Eavenson, that you mean that this equalization of tax would put the two fuels on an economic parity as to their usefulness? And that if they were on such a parity more coal would be used as against oil and gas? If they were on a par would not oil and gas still often be more desirable, considering the convenience involved, as well as the cleanliness and other factors of that sort, and would not the final effect be that the coal industry would not be greatly benefited, whereas the oil and gas industry might be injured greatly, and the net result would not be of any benefit to anybody?

H. N. Eavenson.—The answer is that the real purpose of all this regulation, as far as the coal business is concerned, was purely a movement to help labor. There is about two and a half or three times—we have not been able to get authentic figures of oil and gas production costs—as much labor per unit employed in producing coal as there is in either of the other two fuels. There is no thinking coal man who wants to put the price of either of those fuels on a basis where they cannot be used for the purposes for which they are best fitted.

For instance, nobody would think of trying to prevent the use of natural gas for domestic cooking, or purposes of that kind. There is a very decided feeling that in a market where there is no natural gas convenient and it is transported over a great distance, the people who buy it should be made to pay the reasonable cost of producing and delivering it. If there is any purpose for which natural gas or oil is better adapted or can be used more conveniently than coal, nobody wants to prevent such use. But, they do want to keep the liquid fuels out of markets where coal is better under almost any conditions, when they are anything like equal in price.

A proposition was made when this tax matter came up two years ago, that if natural gas would be sold at the cost of delivery, that is, the actual cost of transportation, plus the 7 or 8¢ a thousand feet it usually costs at the wells this would be entirely satisfactory. When it was proposed to the gas people they said, "No, we are going to sell up to the capacity of the pipe lines first and then the oil and coal people can have what is left."

It is a question of whether you want to put out of work a large number of people. I do not think it would seriously affect the number of men employed in producing either gas or oil, but the competition puts out of business about 150,000 miners. That is the way it affects the coal industry; do we want to put these miners permanently on relief, or try to equalize in some fashion the costs of these fuels on a competitive basis for the purpose for which each one of them is best fitted?

W. A. Forrester, Jr.,* New York, N.Y.—I wonder if this is a fair question? Do you believe that the economic trends whereby there has been a substitution of oil, natural gas and hydro as sources of primary power have more or less run their course in replacing bituminous coal? In other words, have they not made as great inroads into the total fuel markets as would be expected unless there is some drastic change in the relative price levels?

H. N. Eavenson.—It is difficult to tell that because a difference of a few cents a gallon, or a cent a gallon on fuel oil, along the Atlantic seaboard makes the difference between the use of coal or oil. When the price of coal was raised on account of the

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* National City Bank.

Code, there was a very decided loss to oil immediately because the coal was increased 40 or 50¢ a ton on the Atlantic seaboard, and there was a greatly increased market for fuel oil. Then the price of oil was raised and consumers began to go back to coal. Later the price of oil went down. Now it is going up again. If the movement keeps up the plants that are using, and can use, either coal or oil indiscriminately, will of course use the cheapest fuel. There is no question at all that gas and oil will take a lot of fuel business away from coal if the present disparity in prices is maintained.

Makers are selling 300,000 domestic oil burners every year now. They are going into this service at a cost that is a good deal more than that of coal. In other words, almost anybody can heat his house by bituminous coal more cheaply than by either oil or gas. Coal interests are beginning to combat that by putting in the small stokers. You see the effect of this competition in the anthracite industry, whose output decreased 10 per cent last year. I do not think that this competition can be met without a reduction in the price of anthracite.

A. Notman,* New York, N.Y.—Whether or not any of us, or all of us, are personally interested in the fuel industry, we are going to be materially affected by any governmental policies adopted toward those industries. I think that the country as a whole has to make up its mind pretty promptly whether it is a true function of Government to attempt, by legislation, to maintain all existing units of industry on a profitable basis; that is, in order that men employed in those industries at this particular time in our development should continue to be employed. That question has two aspects: first, whether it is a proper function of Government; second, if it is a proper function, whether Government can by regulation accomplish the desired result. The record seems to indicate to my mind at least that it cannot. Yet, we are continually urged to continue to make the effort.

I think that it is just as futile to ask the Government to give us any such guarantee as it would be to ask them to give us a guarantee of long life and happiness individually. I think that is the real issue before us. Much as we may be concerned about the situation of these miners that are no longer needed, there is another side to the picture in that in general prosperity is enhanced by being able to secure necessities at a steadily decreasing cost. Why should we attempt to bolster up units that are no longer necessary?

W. A. Forrester, Jr.—Probably I am not able to give an answer, but we have a Government so we can all live together with a certain degree of security and we have had a breakdown in social security and law and order, in the anthracite field. We have bootlegging and one thing and another, and it has not been possible to protect property rights. It is largely a social problem. How are we going to answer it? I do not consider it good government not to take notice of the problem.

W. O. Hotchkiss.—That brings up a subject of long controversial interest; that is, in regard to the copper industry. Before I went to live in the copper country of Michigan, I appreciated fully the logic of Mr. Notman's position and it took me several years to switch around to some extent. But this thing I found to be true, I think, from consideration of the actual facts as I saw them in living for 10 years in a district that was marginal, is marginal, in production of copper—and I think the same applies to the coal situation—that the pure logic of the situation is unquestionably in favor of Mr. Notman's position, that the interest of the public is best served if we give the public the material it needs, and uses at a minimum price.

On the other hand, our civilization, our institutions, our business is all organized in such a way, and it must necessarily be so, that we cannot instantaneously, we cannot even in a short time, follow the changes that develop with shifts in technologic uses. There is a real necessity for governmental activity, I believe, to serve as a cushion over a fair period of years, which may be as much as a generation for the change from coal or gas—or copper, in northern Michigan, where there is a population of 50,000 people absolutely dependent upon an industry—to provide conditions that will permit of a shift of the occupation and the support of that large group of our population which has a vested interest that we must consider. The Government, I believe, has a proper activity to provide by some means, if possible, a cushion for that transfer to take place without the severe suffering to which many people would be subjected if we followed the strict logic of the situation. I think that is true in connection with our coal-mining situation. We cannot move 100,000 miners, or whatever the number may be, out of the coal industry and into other industries immediately or in a few years. It takes a long time; and in order to prevent the suffering that such an attempt would involve there is a vital function, I believe, for the Government to help tide over that necessary transfer.

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* Consulting Engineer.

I sympathize with the coal people, and I sympathize with the oil people. I sympathize with the railroads, whose business has been cut into most grievously by the truck and the bus. All of those things are transformations that must come. But, I believe we must, if our Government is to serve an essential purpose, help to tide over those things, to stretch them out over a period of time that will entail a minimum of suffering to the many in our population who are involved.

T. T. READ,* New York, N.Y.—One additional point: we must think of these things not simply in terms of the present situation but in terms of the long future. In his oral summary of his paper Mr. Eavenson seemed to me to regard the 15 trillion B.t.u. derived from bituminous coal in 1923 as an all-time peak that the coal industry could never again hope to attain. The decline to two-thirds of that amount in 1934 is indeed discouraging, but our total of B.t.u. hit its low in 1932 and has since been climbing; in 1934 it had regained one-third of its loss since 1923. Bituminous coal by 1934 had regained only one-sixth of its lost ground, but if the total B.t.u. reaches in 1940 the level where it was in 1923 (which it might easily do) bituminous coal will have recovered half of its lost ground and would not need to retrieve very much of the ground it has lost to oil and natural gas to put it back at the 1923 level also. The most reassuring thing about bituminous coal is that we know it is there and we can produce it at any desired rate. What the coal industry needs is the economic equivalent of unemployment insurance to lessen the shock to it of unexpected and rapid temporary declines in coal consumption. How that could practically be done I will not attempt to say.

R. K. WARNER, New Haven, Conn.—I am in entire accord with Dr. Hotchkiss, desire to furnish by law a cushion for this adjustment. We should avoid, however, going to the extreme of attempting to prevent the result of an economic law by passing a law through Congress. In other fields we have tried the latter plan on several occasions entirely without success.

F. E. LATHE,† Ottawa, Ont., Canada.—My first question has to do with statistics that were given at the beginning of the paper comparing the situation of 1920 with that of 1934. Mr. Eavenson first shows that the production of coal has decreased 37 per cent, then goes on to state that the number of miners employed has decreased 28 per cent. That indicates that the output per man per year has decreased appreciably during that time. Further down we find that the output per employee has changed from 4.0 to 4.4 tons, or an increase of 10 per cent. I presume that the apparent discrepancy is accounted for by the fact that the number of days of work per year has decreased substantially during that period?

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† Technical Assistant to the President, National Research Council.

The second question has to do with the transportation of gas or coal to certain areas. Mr. Eavenson has referred to the fact that gas is being delivered in certain areas below the cost of transportation. In previous addresses which I have heard him give he has stated that gas was being sold at a price lower than the freight charges on coal to those areas. In this case he is possibly referring to the cost of transportation of the gas rather than that of the coal?

H. N. Eavenson.—That is correct.

F. E. Lathe.—If that is true, I understand him to ask the gas operators to consider the interest on their investment, to consider that they have the pipe lines to pay for. In that respect is he not taking a different attitude from what he would take in regard to the producers of bituminous coal? The producers of bituminous coal for some years have had to forget their investment. They have tried very hard to make an operating profit. Is it not fair that the gas producers should be allowed to take the same position and consider only their operating profit?

H. N. Eavenson.—All of the figures come from Federal statistics and, of course, they are not based on the same number of working days per year. There is no doubt that a considerable amount of the difference in the increase in the tons per day is caused by the fact that there are so many of the small mines, which are usually the more inefficient ones, that have passed out of the picture and the work is being concentrated in the larger mines.

As far as the gas figures are concerned, I was not so much interested in the interest on investment as I was in the total cost and the depreciation that is charged against that. The average total cost of transportation is 3¢ per M. cu. ft. per 100 miles (Report of Federal Trade Commission 84A, p. 143). In other words, a 1000-mile line to Chicago running at full capacity would cost approximately 30¢ for 1000 ft. of gas delivered there. When it gets down to 20 or 30 per cent capacity, it may cost 70 or 80¢ per thousand feet. That is the way those lines are working now. They are not forgetting anything.

I do not want anybody to think that as a representative of the coal industry I am trying to cry on anybody's shoulder about this. The only thing that the industry wants to know is whether it is proper for it to be handicapped by Federal legislation and its competitors be left alone. That is our attitude primarily.

I think Mr. Notman is right on the logic of the situation. The coal industry does not expect, in fact, I do not think any thinking man would ask, the Government to enable the coal producers to make money on every mine that is operated. I do not think that anybody has any idea that the millennium will come under any such condition. There is no expectation on the part of anybody in the industry that these men now outside shall be put back to work. Of course this is a proposition to preserve labor and living conditions at our mines and that is what the Government intended. The industry did not ask for that. We do not want to have 100,000 or or 150,000 additional men out of employment owing to causes over which the Government has control, and over which we have not, and that is the competition with oil and gas.

As a matter of fact, I do not think that we need to worry about this change, if it is a change, increasing the cost of our fuel, because the whole movement in the industry now—and it is increasing rapidly and is going to increase—is toward mechanization and consolidating the plants in the larger units and running them double shift, which inevitably is going to reduce the price and put many more men out of work.

We have to do it to meet the existing high wage rates and to compete with the other fuels. It cannot be helped. That condition is taking place now rapidly and it is going to increase.

With the existing number of units in operation, there is not enough coal being used to absorb the output. The only way that the mines can be taken care of on an economic basis is to shut down a great many of the inefficient ones and allow the others to run on something like a uniform basis, and in trying to divide up the work as well as it can be done to take care of the men.

The only way that that is going to be done—well, a good many people say that it is the survival of the fittest. I do not know who the fittest are in the coal business. You can see the published reports of 10 or 12 of the largest units in the industry, and only one of them has reported any profits for the last five or six years. The others may be the fit ones to survive, but even the second biggest unit in the industry has been going through a reorganization and is getting ready to start over again. It has wiped out two-thirds of its capital.

The only thing that I can think to do is to consolidate as many of these plants as are fit to survive into larger units where they can be taken care of and the charges absorbed, then shut down many of them and run the remainder at the most economical rate that can be devised. I think that is going to happen whether it is done by law or otherwise.

The industry is not asking for favors, or anything of that kind. I do not know that we could get them if we did. We do think that these handicaps that are put on by the Government, as far as our competitors are concerned, ought to be removed or else they ought to be put on our competitors, too. We do not care which is done.

The other thing that Dr. Read brought out is the fact that a great many people—and I have tried to say this in the last two years—do not appreciate the importance of the fuel industry in the United States. We have heard a great deal of crying in Washington about the silver industry. Ten states every year have produced more coal than all the silver that we have in the United States. You have increased the cost of coal 70 per cent without adding anything to the price. If you increased the price of coal 10 per cent, we should be perfectly willing and think we were sitting on top of the world. And, if the Government would say that it would take all of the coal at that price, all that we can produce, and store it, we would be happy about it.

A. NOTMAN.—I think it is worth while considering, in view of Mr. Eavenson's very apt remarks, that if all these efforts to—How will we put it?—gouge the consumer with successful conspiracies by industry and labor and agriculture, were successful the net result would be just zero internally, but the net result in foreign markets would be that all of our activities would be established on such a high cost and price basis that so far as the outside world is concerned we would be completely shut off. We would have nothing we could trade with them.

Economic and Competitive Poistion of Illinois Coal

By Walter H. Voskuil,* Associate Member A.I.M.E.

(St. Louis Meeting, October, 1935)

Illinois supplies coal to seven states in the Upper Mississippi Valley —Illinois, Missouri, Iowa, Minnesota, Wisconsin, and portions of the Dakotas, Nebraska and Kansas. In this same area are marketed large quantities of coal from West Virginia, Kentucky, Pennsylvania, and elsewhere; fuel oil from the crude petroleum supplies of the Mid-Continent fields, and natural gas from Louisiana, Kansas and the Texas Panhandle. The boundaries of this "Illinois coal-market area" are determined by competition from other coal fields and from other forms of fuel. Within the area so described 90 per cent of Illinois coal is marketed. Thus in the Southwest, an abundance of fuel-oil supplies and natural gas dominates the market almost to the exclusion of coal. The westward movement of Illinois coal in Kansas, Nebraska and the Dakotas is met by an eastward flow of high-grade coals from Colorado, Wyoming and Montana. In the lake-shore counties of Minnesota and Wisconsin, the market is dominated by Appalachian coals, cheaply carried over the Great Lakes and reaching the ports of Lake Michigan and at the head of Lake Superior. Illinois coals find their largest market in the southern and western sections of these two lake states. Only small quantities of Illinois coals are shipped east of the state line. This market area is occupied entirely by the neighboring coal fields of Indiana and the Appalachian coal fields in Ohio, West Virginia, Kentucky and Pennsylvania.

Illinois Coal-market Area

With these factors in mind, the map of the Illinois coal-market area may be drawn. Within this boundary is the area that is logically and economically served by Illinois coal. However, this area is by no means exclusively dependent upon coal from Illinois mines. The preponderance of Appalachian coal in Wisconsin and Minnesota has been mentioned. Appalachian coal, also, moves in large tonnages into Illinois itself, particularly the Chicago district. The Iowa and Missouri local coals share the market with imported coal from Illinois; throughout the market area, fuel oil and natural gas are important factors in the total energy supply.

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TABLE 1.—*Summary of Energy Consumption in the Illinois Coal-market Area in* 1929 (*Exclusive of Gasoline*)

	Quantities	Tons or Equivalent in Tons of Coal
Coal, bituminous, tons		102,858,155
Coke, tons		4,580,764
Briquets, tons		687,377
Anthracite, tons		2,705,946
Fuel oil, barrels	28,871,165	6,880,000
Natural gas, M. cu. ft.	95,410,000	3,816,400
Water power, M. kw-hr.	2,814,435	2,388,000
Total		123,916,642

TABLE 2.—*Bituminous Coal Consumed in Illinois Coal-market Area,* 1917 *and* 1929 (*Exclusive of Railway Fuel, Colliery Fuel and Local Sales*)

	1917		
	Illinois	Other States	Total
Illinois	25,780,675	11,192,221	36,972,896
Wisconsin	1,936,000	6,625,768	8,561,768
Minnesota	1,801,000	4,734,132	6,535,132
Iowa	4,026,000	4,161,697	8,187,697
Missouri	6,806,000	4,477,733	11,283,733
Kansas	107,000	3,554,356	3,661,356
Nebraska	661,000	2,800,842	3,461,842
South Dakota	231,000	847,961	1,078,961
North Dakota	43,000	1,238,347	1,281,347
	1929		
	Illinois	Other States	Total
Illinois	23,148,112	33,704,716	56,852,828
Wisconsin	846,811	8,969,040	9,815,851
Minnesota	767,781	7,718,958	8,486,739
Iowa	2,815,630	5,195,672	8,011,302
Missouri	5,884,713	4,376,632	10,261,345
Kansas	137,299	2,230,686	2,367,985
Nebraska	596,666	3,041,850	3,611,516
South Dakota	172,048	1,000,583	1,172,631
North Dakota	2,799	1,875,159	1,877,958

The consumption of energy materials, exclusive of railway coal and gasoline for automobiles and other uses amounted to an equivalent of 124,000,000 tons in a year of high productivity—1929. Of this amount 103,000,000 tons consisted of bituminous coal and the remainder was distributed among coke, anthracite, fuel oil, briquets, natural gas and water power. See Tables 1 and 2. In order to understand the trend of coal consumption from Illinois fields into this market, it is necessary to analyze: (1) rise of competitive coal fields; (2) growth in supply of fuel oil; (3) rise of natural gas; (4) extent of hydroelectric installation; (5) trends in demand for energy.

The origin of coal consumed in this market area by principal groups of producing fields for the years 1932, 1933 and 1934, is shown in Table 3.

TABLE 3.—*Origin of Coal Shipped into the Illinois Coal-market Area, 1932–1934*

	1932	1933	1934
All-rail shipments			
Western Pennsylvania	2,244	7,653	23,354
Altoona-Somerset-Meyersdale	34,224	62,324	60,582
Fairmont	24,352	17,638	22,755
Northern and Eastern Ohio	3,883	7,206	7,829
Southern Ohio	2,954	2,332	4,561
Kanawha-Logan and Kenova-Thacker	1,415,165	1,335,687	1,269,253
New River-Winding Gulf and Pocahontas-Tug River	7,430,120	7,413,944	7,367,262
Northeast Kentucky and McRoberts	1,247,622	1,385,570	1,445,814
Virginia	98,736	107,430	140,175
Harlan, Hazard	2,173,267	2,235,299	1,896,409
Ex-river coal	1,727	294	248
Northern Illinois	2,162,489	2,186,336	2,798,006
Central and southern Illinois	15,476,687	15,860,079	17,161,240
Indiana	4,690,607	4,571,424	4,874,352
Western Kentucky	3,855,747	2,233,645	1,908,654
Lake cargo coal	13,287,000	17,176,000	17,487,000
Local production			
Iowa	3,862,000	3,195,000	3,345,000
Missouri	4,070,000	3,432,000	
Kansas	1,953,000	2,218,000	5,800,000
North Dakota	1,740,000	1,841,000	1,830,000

The significant feature of this table is the large shipment of coal from the southern West Virginia and eastern Kentucky fields. The present important position of the southern Appalachian group of fields in the Mississippi Valley market has its origin in the factors and events that promoted the exploitation of these fields faster than the general rate of increase of coal consumption in the United States. The trend of production in the southern West Virginia, Virginia, and eastern Kentucky fields is shown in Table 4.

TABLE 4.—*Coal Production in the United States and Lower Appalachian Fields*
Millions of Net Tons

Year	Total Coal	Lower Appalachian Production	Percentage of U. S. Production
1909	461	61	13.2
1910	502	75	14.9
1911	496	74	14.9
1912	534	83	15.6
1913	570	91	16.0
1914	514	92	17.9
1915	532	99	18.6
1916	590	114	19.3
1917	651	114	17.5
1918	678	121	17.8
1919	554	110	19.9
1920	659	126	19.1
1921	507	103	20.3
1922	477	120	25.2
1923	658	154	23.4
1924	572	149	26.0
1925	582	178	30.6
1926	658	205	31.2
1927	598	206	34.4
1928	576	191	33.2
1929	609	198	32.6
1930	532	172	32.3
1931	442	143	32.4
1932	360	119	33.1
1933	383	131	34.2

The influence of rail rates on coal to lower lake ports, and of lower wage rates in the southerly fields in promoting development in this area need not be reviewed here; the story is too long and too intricate. Nevertheless, the net effect of these two factors was to increase the coal-producing capacity in a district that had only a very limited local market. For this group of mines in West Virginia, the outlet must be eastward to the Atlantic seaboard markets or westward to the Mississippi Valley; the near-by market of western Pennsylvania and eastern Ohio is dominated by the Pennsylvania and northern West Virginia fields. In maintaining and enlarging their distant markets, the district has been aided by the freight structure. For example, the rate from the New River field (West Virginia) to Hampton Roads is $2.25 per net ton while the rate from Clearfield (Pennsylvania) to New York City is $2.38 per net ton, although the distance is 67 miles less. Similarly, the rate from New River to Toledo is $2.04 per net ton over a distance of 436 miles while the rate from Clearfield to Cleveland is $2.31 a ton for a distance of 302 miles. The all-rail rate westward also favors the southern group of mines. Differences in wage rates between southern and northern fields also make their contribution to the enlarging market outlet of the former fields. The non-union fields of the South, favored with better coal, operating under flexible wage scales readily adjusted to market conditions, free to sell coal during shutdowns in the union fields, and subsidized in part by a lower ton-mile freight rate in coal, gradually displaced the northern fields in the distant markets. As railroad facilities increased, the newer sections of the non-union areas expanded rapidly.

The large shipments of coal into the Illinois coal-market area, however, are made in the face of a higher total freight rate than shipments to similar market points from Illinois fields. In 1929, for example, these southern Appalachian fields shipped 24,000,000 tons to this market area. The distribution of so large a quantity of coal is accomplished therefore upon factors other than transportation advantage. Some of this is due to demand for coal of special quality, as, for example, coking coals, or smokeless coals. Some of the coals find a preferential market among domestic consumers, public utilities and railroads.

Lake Cargo Coal

Tonnages of coal carried over the lakes to Lake Michigan and Lake Superior ports have varied in recent years from 13,000,000 to 17,000,000 tons. Of the southern fields, the Kanawha district is the leading source of supply, Pocahontas is second, and Kenova-Thacker is rapidly assuming an important position.

The markets for lake coal are concentrated mainly in Wisconsin and Minnesota. The advantage of low-cost transportation into this part of the market is so pronounced that Illinois coal producers will find difficulty in overcoming the handicap of all-rail haul from the southern part of the state.

Fuel Oil

In order to evaluate the competitive position of fuel oil in the energy market, it is necessary that the economic nature of this product be clearly understood. Fuel oil is a byproduct of the gasoline-manufacturing industry. As a fuel it serves no peculiar need (with minor exceptions) that cannot also be supplied by coal. The large supply of fuel oil available in the United States is due to two factors: (1) The nature of oil and the methods of refining were such as to yield a larger percentage of the byproduct fuel oil than the cash crop of the refining industry—gasoline. Until 1925, 50 per cent or more of the products of crude oil was in the form of fuel oil. (2) The mounting demand for gasoline occasioned by the rapid expansion of automobile output was accompanied by the production of correspondingly large quantities of fuel oil. In 1934, the deliveries of fuel oil amounted to 332,286,000 bbl., an equivalent of 80,000,000 tons of coal. Although the wider use of cracking methods is effecting a higher percentage of gasoline recovery, this has not brought about an appreciable decline in the available supply of fuel oil. The key to the quantity of fuel oil is the rate of production of crude oil. As long as crude oil is being produced abundantly, the refining industry finds it expedient to recover from the crude only the percentage of gasoline that can be obtained by skimming and cracking methods and to throw the residue on the market as fuel oil at whatever price it will bring.

Conditions Governing the Distribution of Fuel Oil

Since it is difficult or impossible to transport fuel oil by pipe line and it is costly to carry it by tank car, the key factor in determining the major areas of consumption is refinery location. In the Illinois coal-market areas four well defined refinery centers are: (1) the Chicago district, (2) the East St. Louis district, (3) Eastern Illinois district, and (4) Kansas City.

The distance over which fuel oil can be transported from the refinery center to the consuming market is determined largely by the rate at which the freight charges bring the cost of fuel oil up to a competitive level with coal. A secondary factor that may enlarge somewhat the radius over which fuel oil is moved is demand occasioned by special uses or convenience. The concentration of fuel-oil utilization near the refinery centers is illustrated by data for consumption of fuel oil in manufacturing as revealed by the Census for 1929. In that year the consumption of fuel oil reported for all uses[1] in the Illinois coal-market area was: entire area, 28,871,165 bbl., exclusive of Nebraska and Kansas, 22,343,644 bbl. Of this amount, the Census on Consumption of Fuel and Electric Energy in Manufacturing Industries allots to manufacturing the following amounts: entire area, 20,353,200 bbl., or 70 per cent of the total consumed; exclusive of Nebraska and Kansas, 16,758,188, bbl., or 75 per cent of the total consumed. The degree to which this consumption is concentrated near the source of production is revealed in Tables 5 and 6. Next in importance to the refinery centers as consumers of fuel are the urbanized areas within these states.

Fuel oil consumed for manufacturing in the remaining counties of these states, after deducting the quantities used near the refining centers and the somewhat smaller quantities consumed in urban centers not located near refineries, amounts to 1,306,200 bbl. (in the area exclusive of Kansas and Nebraska) or 7.8 per cent of the total used in manufacturing. Although this distribution accounts for only the fuel oil used in manufacturing, and does not include fuel oil used for domestic and commercial heating and railroads, nevertheless it represents 75 per cent of the total consumption and is a fair indicator of the geographical distribution of this type of fuel.

This analysis implies that fuel oil is consumed near the sources of production but takes no account of the balance between production and

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[1] U. S. Bur. Mines: Report on National Survey of Fuel Oil Distribution (1929).

TABLE 5.—*Fuel Oil Used by Illinois Manufacturing Industries in Illinois Coal-market Area*
Near Refining Centers

Refining Center	Consuming Counties in	Barrels
Chicago	Wisconsin	1,572,530
	Illinois	5,611,815
	Indiana	6,083,004
	Total	13,267,349
East St. Louis	Illinois	3,536,634
	Missouri	930,000
	Total	4,466,634
Eastern Illinois	Crawford County Lawrence County	685,650
Kansas City St. Joseph	Missouri	1,091,572
	Total	19,511,205

TABLE 6.—*Consumption in Principal Localities*

Area or City	State	Barrels
Tri-City district (Rock Island, Moline, Davenport)	Illinois	274,423
	Iowa	159,552
	Total	433,975
Peoria	Illinois	519,190
Danville–Champaign	Illinois	76,822
Joplin district	Missouri	23,262
St. Paul–Minneapolis area	Minnesota	596,482
Adrian County	Missouri	95,108
Henry County		88,988
Duluth, St. Louis County	Minnesota	58,925
Crown Wing County	Minnesota	26,428
	Total	2,023,787

consumption within major refining zones. The situation in the Illinois coal-market area is presented in Table 7, which gives production and consumption of fuel in the states comprising the Illinois coal market proper, and the same data for those states southwest of the Illinois coal-market area in the Mid-Continent field of which the principal refining area centers around Tulsa, Oklahoma. Indiana is included with the first group because of its large refining capacity near Chicago and a consequent outlet of its fuel oil into the Chicago area. The excess of production over consumption necessitates importation from the Mid-Continent field, where the opposite condition exists. The transportation of fuel oil over this long distance was no doubt made economically feasible by the low prices prevailing in 1929, aided to some extent by cheap barge transportation over part of the distance.

TABLE 7.—*Consumption and Production of Fuel Oil*, 1929
BARRELS OF 42 GALLONS EACH

	Production	Consumption	Excess
Illinois	7,618,148	13,257,751	Deficiency
Indiana	11,327,427	5,581,087	
Wisconsin	0	1,640,396	
Minnesota	0	1,548,860	
North Dakota	0	109,655	
South Dakota	0	154,290	
Iowa	(Included with Indiana)	881,970	
Missouri	(Included with Indiana)	4,750,722	
Nebraska	0	810,027	
	18,945,575	28,734,758	− 9,789,183
Arkansas	5,764,696	2,633,170	
Kansas	8,177,175	5,717,494	
Oklahoma	21,455,321	11,971,557	
	35,397,192	20,322,221	+15,076,971

The economic nature of this fuel as a byproduct of the gasoline-manufacturing industry and a fuel market of such a nature that an exclusive outlet does not exist compels the refiners to sell it at prices determined by the price of coal. An excess of the fuel over normal needs, both in Chicago and the Mid-Continent, has the effect of further depressing prices below the level of an equivalent quantity of coal, with the consequence that long-distance shipments become feasible.

The trend of fuel-oil supply and prices will depend upon future conditions of crude-oil production. The current potential supply in excess of effective demand is largely the result of the discovery, almost simultaneously, of several prolific pools, of which East Texas, discovered in 1930, is probably the largest. Since that year, there have been no major pool discoveries. In this connection, it may be interesting to examine a table prepared by Wallace E. Pratt, vice president of Humble Oil and Refining Co., on the production and discovery experience of the American oil-producing industry by periods (Table 8). Of particular significance are the data presented in columns 4 and 5 of this table. Consumption has been gradually gaining upon new discoveries until it has run from 0.3 to 0.6 of total discoveries, and in the period 1931–1934 consumption has been 50 per cent greater than discoveries. Unless a cycle of prolific discoveries recurs, as in the 1926–1930 period, the supply of crude oil will begin to decline or will be maintained by the more costly methods of deeper drilling and the use of improved recovery methods in old fields. In either case, there will be a tendency to convert a larger portion of the crude oil into gasoline and thereby reduce the available fuel-oil supply.

TABLE 8.—*Analysis of Production and Discovery Experience of the American Oil-producing Industry, by Periods*

1 Periods Analyzed	2 Average Annual Production, Millions of Barrels	3 Average Discoveries, Millions of Barrels	4 Ratio of Accumulated Production to Accumulated Discoveries	5 Ratio of Annual Production to Average Annual Discoveries	6 Ratio of Average Annual Discoveries to Total Discoveries for All Periods
1859–1900	25	80	0.3	0.3	0.003
1901–1905	105	340	0.3	0.3	0.01
1906–1910	175	275	0.4	0.6	0.01
1911–1915	250	500	0.4	0.5	0.02
1916–1920	370	585	0.5	0.6	0.02
1921–1925	650	820	0.6	0.8	0.03
1926–1930	895	1990	0.5	0.4	0.07
1931–1934	870	580	0.6	1.5	0.02

NATURAL GAS

Natural gas from the Mid-Continent field became a factor of increasing importance in the Illinois coal-market area after 1928, when the first long-distance pipe lines for gas began to invade the area. Importation of gas has risen from 9406 million cubic feet in 1928 to 80,578 million cubic feet in 1933 (into the states of Illinois, Iowa, Missouri, South Dakota and Nebraska). In 1934 the gas line into Minnesota was put into service. Consumption of natural gas in the Illinois coal-market area from 1928 to 1933 is shown in Table 9.

The record of increasing importation and consumption of natural gas in the Illinois coal-market area raises the question as to the ultimate extent of utilization of this fuel and the effect of its introduction upon the competitive energy market. The answer to this question resolves itself into an analysis of potential supply of gas, extent and location of probable market outlets and the price at which this fuel can be delivered in the markets of the Upper Mississippi Valley.

Natural gas available to the Mississippi Valley region is produced in abundant quantities in the Mid-Continent, Gulf and Rocky Mountain fields, in limited quantities in the states of Kentucky, Tennessee, Indiana, Illinois and Ohio, both as a byproduct from oil wells and from fields that are primarily gas producers. The producers and consumers of energy materials in Illinois are particularly interested in conditions of supply and demand in Kansas, Louisiana, Oklahoma and Texas, from which states the natural gas consumed in the Illinois coal-market area is obtained. In 1933, these four states operated 8810 gas wells and 126,650 oil wells, the majority of which produce enough gas to justify the efforts to save it. Production of gas in these states has increased enormously and, although consumption for field, industrial and domestic use showed corresponding increases, surplus gas available for export became more abundant. The comparative position of production, consumption and surplus for these states is shown in Table 10.

TABLE 9.—*Consumption of Natural Gas in the Illinois Coal Market Area* 1928–33
MILLIONS OF CUBIC FEET

	1928	1929	1930	1931	1932	1933	1934
Illinois	3,051	3,139	9,602	14,050	29,432	33,377	45,084
Iowa	0	0	0	3,522	7,533	11,408	16,636
Missouri	9,776	15,078	26,122	24,261	25,310	27,579	29,792
South Dakota	214	1,717	2,905	2,803	2,776	3,264	3,901
Nebraska	0	0	1,098	4,817	8,661	10,293	12,789
Total	13,031	19,934	39,727	49,453	73,712	85,921	108,202
Approximate coal equivalent (tons)	521,240	796,360	1,489,080	1,978,120	2,948,480	3,436,840	4,032,810

Table 10 discloses some interesting and significant characteristics in the natural-gas industry of these states. Consumption has failed to keep pace with production, with the result that in the 15-year period the surplus exportable gas has increased elevenfold. Consumption by uses within the area shows a rapid gain in industrial consumption, a gain proportional to production in field consumption, and a slower increase in domestic use. The question of interest is to what extent the local market can be expected to absorb the available supplies of natural gas and how much is available for export. One examination of trends in individual items of consumption may throw some light upon what may be expected in the immediate future.

Field use (for drilling, pumping, and operating gasoline-recovery plants) has increased from 105,163,000 to 445,274,000 cu. ft., or a ratio of 1:3.2. In the meantime, crude production in these states has increased from 287,091,000 bbl. to 652,004,000 bbl., or a ratio of 1:2.3. Consumption of natural gas for field use has increased therefore more than the proportional output of crude petroleum. This may be due to the substitution of gas for oil engines in field operations, increased pumping from wells and through pipe lines. That this increase will continue is doubtful in view of the changing conditions in the oil fields. Production, which reached a peak in 1929, declined through proration agreements and curtailment of operations. Production in 1935 will probably exceed that in 1934. Gasoline demand apparently has reached a stabilized level, for the near future at least, a factor that will act to restrain an increased demand for crude oil. Increased recovery of gasoline from a barrel of oil will further retard the demand for crude. Accompanying a stabilized output of crude oil will be stationary consumption of natural gas in oil-field operations.

TABLE 10.—*Production, Consumption and Surplus of Natural Gas from Kansas, Louisiana, Oklahoma and Texas*

	Millions of Cubic Feet		Per Cent	
	1920	1934	1920	1934
Production	270,962	1,130,055	100.0	100.0
Consumption	255,515	953,780	94.3	84.1
Surplus	15,447	176,275	5.7	15.9
Consumption, by uses:				
Domestic	47,816	87,641	18.7	9.2
Industrial	102,536	468,112	40.2	49.1
Field	105,163	398,027	41.1	41.7
Total	255,515	953,780	100.0	100.0
Industrial uses:				
Public utility	9,564	71,260	9.3	15.2
Carbon black	18,100	222,535	17.7	47.5
Petroleum refineries		45,786		9.8
Other industrial	74,872	128,531	73.0	27.5
Total	102,536	468,112	100.0	

The rapid rise in industrial consumption also gives indications of deceleration when individual items of industrial use are examined. The most important of these in recent years has been carbon black. Consumption of natural gas, in these four states, in the manufacture of this product increased from 18,100,000 cu. ft. in 1920 to 179,096,000 cu. ft. in 1933. In that year the carbon-black industry absorbed 43.7 per cent of the total industrial consumption. With the saturation of the carbon-black market and a probable shift of this industry to other newly discovered gas fields far from markets, the carbon-black industry will tend to decline in the Mid-Continent field.

The third item in the consumption of natural gas is the electric utility market. The rapid rise between 1920 and 1934 and the probable stabilization of consumption in this field is indicated by the data in Table 11.

TABLE 11.—*Consumption of Fuel by Electric Utilities in Kansas, Louisiana, Oklahoma and Texas*

Year	1920	1930	1931	1934
Coal, tons	961,124	1,116,316	935,444	955,001
Fuel oil, bbl	4,559,047	1,382,290	835,148	742,092
Natural gas, M. cu. ft	9,563,545	81,755,512	74,889,977	70,259,589

The natural-gas market in the states from which the Illinois coal-market area obtains its supply of gas appears to be reaching the point of saturation. Nearly all cities and communities of 5000 inhabitants or more are now connected with natural-gas lines. Consumption for domestic purposes has shown a considerable increase since 1927 but appears to be reaching a point of stabilization. Rapid increases in consumption for carbon-black manufacture are the result of the combined factors of a decline of the industry in West Virginia, a pronounced growth in automobile-tire manufacture in the past decade and in an output far in excess of market demand, especially in 1929 and 1930. Until surplus stocks of this material are sold and further production is necessary, this industry will probably be severely curtailed. When production is resumed, it will probably occur in states that have abundant supplies of gas without adequate markets for disposal through pipe line connections. Future production may be attempted in Wyoming, Montana, and Alberta, Canada. A program of curtailment in the crude-oil industry will have a corresponding effect upon the consumption of natural gas for field operations. With a decrease in the rate of growth of gas consumption, the question of disposal of surplus gas becomes more critical. For this disposal six long-distance outlets are provided, four of which enter the Illinois coal-market area.

CONCLUSION

The future market trends for coal from Illinois in the seven states of the Upper Mississippi Valley depends upon three factors:

1. The trend in demand for fuel.
2. The trend of coal shipments from Appalachian fields.
3. Trends in oil and gas production and marketing.

Although the demand for coal no doubt will recover somewhat from the low years of the depression, a return to the peak of former years is being offset by increased efficiency of utilization in public utilities, in railroad use and in manufacturing operations.

Coal shipments into the Illinois coal-market area from the Appalachian fields, which have increased steadily over the lakes and which have fluctuated somewhat irregularly by all-rail haul, may be adversely affected by wage-rate increases and price regulations, if validity of such regulation is sustained in the courts.

The competition of fuel oil is likely to become less critical as the problem of finding a new supply of oil equal to the annual demand becomes more costly.

The possibilities of further inroads of natural gas exist in the face of a supply in the Mid-Continent area far in excess of local demand.

Acknowledgments

The data in the tables of this paper were drawn from reports of the Bureau of the Census and the U. S. Bureau of Mines. Table 8 appeared originally in the *Oil and Gas Journal* of May 2, 1935.

The Economics of the Distribution of Anthracite

By Norman F. Patton*

(New York Meeting, February, 1935)

The subject assigned is so broad that thorough discussion is well-nigh impossible within the space allotted, and further, few specific data are available upon which to predicate conclusions concerning the efficiency of present methods of distributing anthracite. This paper treats briefly the various agencies through which both primary and secondary distribution are effected, the methods of transportation and the general interrelation of production, demand, inventories and price.

Avenues of Distribution

Three distinct types of primary distribution are common in the marketing of anthracite. First, though not most important, is the sale by the producer, or operator, direct to the consumer. These sales, other than to employees and the local trade, are generally confined to steam sizes for industrial uses, and are consigned to the sidings or docks of the consumer. Sales of this type often are so closely priced to compete with oil and bituminous that the addition of even a small dealer's margin is sufficient to increase the cost to the point where the business might be lost.

Second, the operator sells to a wholesaler or broker for subsequent resale, either to a retail dealer for clearance through his yard or direct to large consumers. Usually this method of distribution is employed by the smaller operators, who do not find it economically feasible to maintain extensive sales organizations. In most instances, the wholesaler retains some 5 per cent for his services, paid, of course, by the producing company whose anthracite he handles. He may, and frequently does, handle the output of several different companies, suited to the varying needs of the different communities he serves. His business includes all sizes of anthracite.

The third, and by far the most general method, consists in selling to retail dealers, shipments being made direct from the breakers to the yards of the dealers. In large consuming areas, such as the metropolitan districts of New York or Philadelphia, one operator may sell to several

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dealers, and generally his sales are spread over wide territories. This type of distribution includes the marketing of all sizes, and is used by large and small operators alike.

Secondary distribution is effected by the wholesalers, in instances in which they resell to retail dealers, and naturally, in transactions of this kind, there occurs what amounts to a tertiary distribution in disposition of the tonnage to the ultimate consumer. Because of the preponderance of the sales by operators to retail dealers, however, it is logical to expect the great bulk of secondary distribution to lie in sales from dealers' yards direct to the bins of the consumers.

A word at this point concerning the areas in which anthracite finds its markets should be enlightening. Ten years ago, approximately 8,300,000 tons annually were shipped to points west of Buffalo. At present, the tonnage to these same points is only about 1,200,000, representing a loss of some 85.5 per cent during the past decade. We need not look far to find causes leading to this disheartening decline. The piping of natural gas to many points formerly served by anthracite, the laying of an ever-increasing mileage of oil pipe lines, the erection of numerous byproduct coke plants, and the freight differentials in favor of bituminous coals not only from such producing areas as Franklin County, Illinois, but from the low-volatile fields in West Virginia as well, have placed anthracite under such a disadvantage that losses have been inevitable.

Nor have losses been confined to the middle west. Importations of foreign anthracites, notably of Welsh, Scotch, German and Russian origin, have grown during this 10-yr. period by the approximate amount of 1,800,000 tons in our New England and Canadian markets. Fuel oil, in large part water-borne, has attracted many thousands of former anthracite users in New England, New York, Pennsylvania and other seaboard communities. Coke from producing units in or near our principal consuming markets has detracted millions of tons from our eastern and New England sales, and natural gas is now being produced in New York State. Our markets have thus been steadily restricted both geographically and in size over the period of the past 10 years. At present, about 79 per cent of the tonnage of anthracite is sold in the three states of New York, New Jersey and Pennsylvania. An additional 12 per cent goes to New England, and 3 per cent finds its way to Canada, leaving only 6 per cent in Delaware, Maryland, the District of Columbia, Virginia and the Middle West.

Nor is this distribution by any means uniform for all sizes. Buckwheat, for example, finds its chief outlets in Philadelphia and New York and in similar points to which freight rates for short hauls are relatively low in relation to other costs. By far the greatest demand in New England, Canada and points in the Middle West where we still retain any tonnage at all is for chestnut and larger sizes. Pea size appears to find its largest market relative to total consumption in Pennsylvania and New Jersey. Naturally, such unequal distribution of sizes sets up a whole series of distribution problems, involving differing mine prices, freight rates and dealers' margins as components in establishing delivered prices in competition with locally favored substitute fuels.

Production, Demand, Price and Inventories

Any study of distribution must of necessity take into account the relationship between the four major factors involved in the balance of an industry's economic structure. These may be enumerated as production, demand, price and inventories. The anthracite situation is complicated by uneven market demands for each of the various sizes, and these varying demands are met by different agencies of distribution. There exists a further complication in the fluctuating seasonal demand for the various sizes with resultant accumulation or relative shortage of inventories, which must be held in check through control of production. Still further, while the prices of pea and larger sizes are usually reduced in the spring, and gradually increased through the summer, prices of buckwheat and smaller sizes do not vary during the year.

First, let us examine the market for steam sizes. While a considerable percentage of the buckwheats finds its way into residences for use in automatic devices such as magazine-feed boilers, stokers, etc., their greatest use is in large buildings, such as hotels, theaters, schools and apartment houses, for heating and power generation, and in no small number of factories. In many instances, the fuel is also utilized in the heating of service water for such structures. Railways and power plants, too, still use considerable quantities of sizes smaller than buckwheat. Consumption is more constant, therefore, and less dependent on weather conditions than in the case of the domestic sizes. The purchasing of steam sizes does not follow their use in a uniform manner; in fact, there are pronounced peaks of demand.

Domestic sizes, on the contrary, are used almost exclusively for heating residences, for heating service water and for cooking, and the demand, therefore, is sensitive to changes in weather conditions and evinces wide variations. This tendency has been accentuated in recent years by the general hand-to-mouth buying on the part of the consuming public in fuel as well as in other commodities. It is necessary, therefore, to maintain a supply of these sizes sufficient to meet such sudden demands as occurred last February, when relatively large quantities of anthracite were moved on receipt of immediate orders.

With such conditions facing the industry, it is easy to see the fine balance needed in mine operation. But here again we encounter a difficult situation. In the handling of run of mine anthracite through the breakers, it is not possible to regulate, except within small limitations, the percentages of steam sizes that accompany the output of domestic sizes. It frequently happens, therefore, that in order to meet demands for one class of sizes, the breakers are operated at a rate that results in the accumulation of unwieldy tonnages of the other sizes. These temporarily unwanted products must be removed from the chutes in order to allow free loading and shipping of those in demand, with the result that at times cars are filled to the extent that storage tracks are utilized to capacity, and car movement becomes a matter of immediate necessity. This, in turn, results in cars being forwarded on consignment, demurrage is frequently incurred, and reductions from circular prices are necessary in order to dispose of the surplus tonnage at a time when the market is not normally prepared to absorb the excess.

Production is also governed in off-season months by the economic necessity of keeping a large portion, at least, of the productive capacity in operation, in order to avoid the costs incident to shutting down and subsequent reopening; and further, by the attempt on the part of the operators to equalize working hours and payrolls over as long a season as possible. The inactivity of the producing end of the industry, therefore, is not as marked during the summer months as might be expected.

In order to stimulate buying activity during the off-season, and also to avoid car congestion occasioned by excessive storage, it has been the practice of the industry for years to place in effect spring reductions in the mine prices of the domestic sizes. But general economic changes incident to the depression and the collateral purchasing power of the communities using anthracite have militated against the former habit of laying in the winter's supply of fuel in the summer months. The average householder is afraid that the mortgage on his home may be foreclosed, and there are so many demands in the spring for his ready funds that he is loath to invest sizeable sums in a commodity which he will not even begin to use for six months or longer, and by spring he is tired of the thought of fuel and its cost, anyway.

Then, too, the anthracite industry has been so free from serious labor difficulties of late that the remembrance of the shortage occasioned by the strike in 1926 has lost much of its persuasion. Six long, warm winters in succession left the public in the frame of mind where they had forgotten what fuel demands might arise from more normal weather. Even last year was only moderately colder than normal in point of degree-day demand for heat, and the operators, carriers and dealers were in position to deliver sufficient amounts of anthracite on little or no notice to tide over the emergency, notwithstanding frozen breakers, switch points, harbors and streets. The threatened "fuel shortage" publicized in the press was—in the case of anthracite, at least—due far more to the placing of orders larger than immediate requirements than to any failure of the industry to satisfy actual demands.

But possibly in the last analysis the failure of the householder to respond to spring reductions in price may be attributed to the inadequacy of these reductions. His winter's fuel bill will amount, let us say, to $120. In April, usually, there is a reduction of $1 per ton on the domestic sizes at the mines. The local dealer may allow a further small reduction, which is the exception rather than the rule, but even if he does, the rails make no freight-rate reductions in the summer, and the net saving amounts to about $15. By the first of January the householder will still have over half of his anthracite unused, he has incurred carrying charges aggregating over $4, and the net economy will approximate less than $10 over the whole winter.

Despite the many complications that beset the industry as respects production and demand for different sizes, with varying inventories and irregular price conditions, it is possible, through the use of indices of seasonal variation, to establish a rather accurate trend of the cycle of monthly changes due to seasonal variations. Far from being straight averages, these indices are predicated upon actual conditions, which have existed each month during the past five years as respects each of seven principal sizes, and are computed in accordance with the statistically correct link-chain method of determining seasonal variations. The indices establish the relation of any month to any other, and, by dividing actual results by the correct indices, permit adjustment for the normal seasonal variations. The curve of the figures so obtained reflects conditions that are attributable to causes other than seasonal.

Rather than analyze each of the seven sizes individually, concerning which data have been computed, let us consider the industry as a whole. In Fig. 1 are drawn to scale the indices of seasonal variation of production, movement, inventories and prices. The numbers also are set forth on the chart. The upper section makes it apparent that production holds relatively firm during the first four months of the year, but that in May curtailment sets in and continues until September. Sharp peaks occur in October and December. November is marked by a distinct decline, owing to the many holidays in that month.

Movement, by which is meant production plus withdrawals (or minus increases) of storage reserves, constitutes the most accurate basis for measuring the takings of the dealers from the producers. It is, however, but little indication of the passage of anthracite to consumers' bins. The demand curve will be seen to follow in a general way that of production. Periods of overproduction are observed from April to October, inclusive, while production falls short of demand from November through March. It will be further observed that production is relatively less subject to change than demand, although both have wide swings. The low point in production is attained in June, when the index stands at 83.4 per cent. In October, the high month, the index reaches 126.1 per cent. This increase of 51.2 per cent in the rate of productive activity is exceeded by an increase of 59.7 per cent in demand, which rises from the June low of 78.1 per cent to the October high of 124.7 per cent.

Overproduction in the spring and summer months and the reverse during the heating season results, of course, in striking variations in storage reserves, as disclosed in the lower half of the chart, in which are plotted indices of seasonal variation in inventories (storage) and in prices. The heavy demands of the early months of the year cause drains on inventories to supply the production deficiency, following which the converse builds up storage stocks to a peak just before the heating season again gets under way. The swings in this curve are much more pronounced than in either production or demand, and indicate an increase of 126.6 per cent from the low of 62.4 per cent in April to the peak of

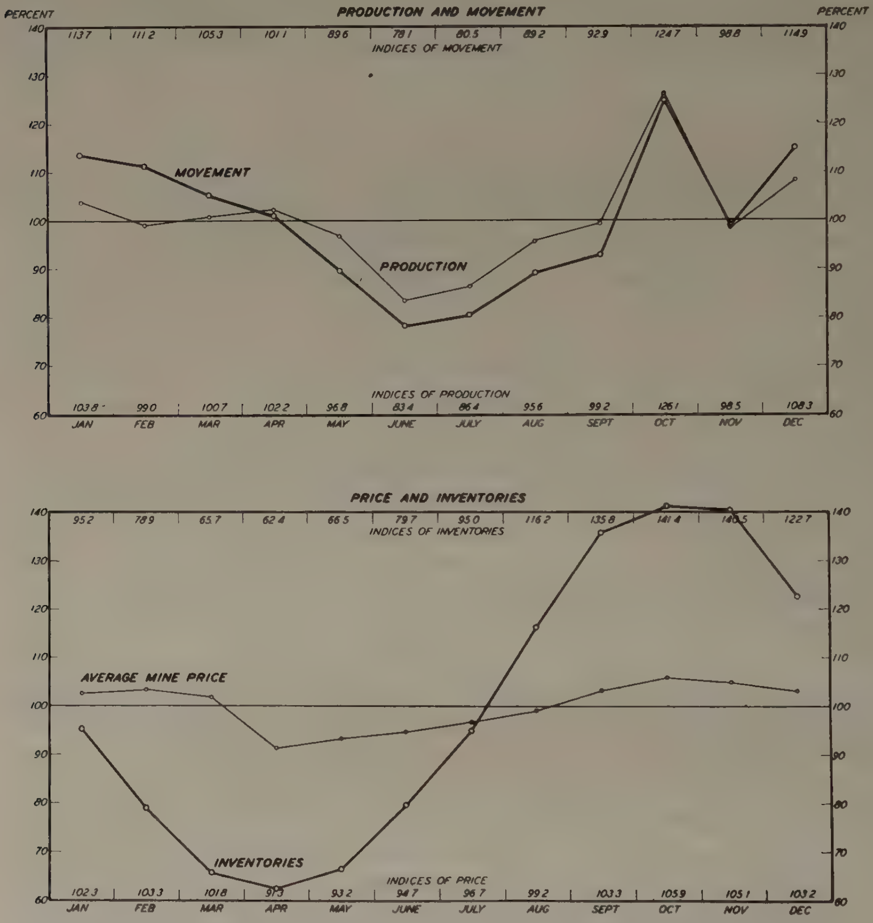

FIG. 1.—INDICES OF SEASONAL VARIATION IN ALL SIZES OF ANTHRACITE.

141.4 per cent in October. In the steam sizes, the variation between the extremes is even more marked.

One of the strongest arguments in favor of anthracite either as a domestic or an industrial fuel is its price stability relative to its competitors. While we find bituminous coal and oil in the industrial field fluctuating radically over a period of time, and even from month to month, with attendant variations in costs to the detriment of the users, and the same condition to a lesser degree in regard to oil and coke in domestic utilization, anthracite is by far the most stable in price of any of the generally used fuels. This has been true over a period of years, and it holds as well during successive months within the year. This is strikingly brought out by the curve of indices of seasonal variation in price. With extreme increases of 51 per cent in production, 60 per cent in demand and 127 per cent in storage reserves from their respective low points, the factor of price shows but a 16 per cent gain from the low of 91.3 per cent in April to the high of 105.9 per cent in October. Surprise may be expressed as to the more or less steady decrease in the price index from October, when winter circular prices are in full effect, through until April. This is explained by the increased sales of the smaller sizes at lower prices, in proportion to sales of the larger sizes at higher prices.

The decline in the price curve coming at the same time as the decline in that of demand makes it appear that reductions have no pronounced effect on movement, nor do they appear to have any marked effect on reducing storage reserves, and it becomes evident that the bulk of sales occurs on a rising, not a falling, price structure.

TRANSPORTATION

Turning now to methods of transporting anthracite from the mines or breakers to the dealers' yards and to the consumers' bins, we find three distinct media. First, and most important of course, is that offered by the railroads direct to trestles or sidings of dealers. No question exists but that high freight rates have contributed in no small degree to the loss of our markets both distant and near by. When we consider that anthracite mined in Wales may be delivered at points in Canada and New England at total freight charges from the mine mouth to the dock of between 45 and 55 per cent of our freight rate from Scranton to the same points; that oil moves almost entirely from the producing fields to refineries through pipe lines or in ocean vessels, is subsequently moved by truck or inland waterways to local distributing points, and thence trucked to the consumers' tanks; that coke plants have been erected in 13 states, within easy trucking distance of many consuming centers, often direct to householders' cellars; that natural gas is piped great distances and then distributed, as is manufactured or mixed gas, through mains laid under city streets; and that none of these substitute fuels may be hampered by rail freight tariffs in reaching their markets, it must be conceded that anthracite, confined to some 490 square miles of mountainous country in inland Pennsylvania, is placed under a severe handicap by the present rate structure. With the exception of a few relatively unimportant reductions in freight rates in 1933, there has been no relief from this item of marketing cost, and the carriers at this writing are defending before the Interstate Commerce Commission their petition for increases which, in the case of anthracite, will amount to 30¢ per net ton in every one of our principal consuming markets.

Naturally, the location of the mines with respect to the various rail facilities determines to quite an extent the areas in which best outlets are to be found. For example, mines on the Delaware and Hudson Railroad enjoy better service to Albany and northwestern New England points than do those on the Reading road, while operations on the lines of the Pennsylvania Railroad have a direct route to Washington. Such situations lead to further unequal distribution of anthracite with respect to different areas.

The second major outlet lies in a combination of rail and water routes, such as to New England and Long Island points over New York and Philadelphia piers, and to many points in Canada and the northern states reached by vessel from such harbor dumps as exist at Buffalo, Erie and other Lake ports. The relatively low combination rates for this service, as compared with all-rail shipments, undoubtedly have prevented an even greater loss in the West and in Canada than we have actually experienced, and enable us to sell, even at present, substantial tonnages in such cities as Milwaukee and Chicago.

The third mode of transportation for anthracite, and one that is receiving the very serious attention of operators, carriers, dealers and governmental agencies alike, is that by truck. Here again, we must differentiate this time between the various types of trucking. We may first eliminate what are known as "local sales." This represents the movement of anthracite to communities within the producing region itself. It is a practice of long standing, accepted by carriers and dealers alike as noncompetitive, and at present amounts to about 3,000,000 tons per year. Next, is the trucking from the mines either to yards of more distant dealers, or direct to ultimate consumers, in direct competition with the railroads or dealers, or both. It is estimated that this movement of anthracite, formerly rail-borne, has more than tripled in the past two years, and now amounts to some 3,000,000 tons per year.

The third form of trucking, and by far the most vicious, is that of anthracite that has been stolen from the lands of the coal-producing companies, trucked to market, and sold to the consumer at prices much lower than those charged by regular equipped dealers whose minimums are determined by freight rates and required code margins. This trucking of stolen coal which, it is estimated, amounts to 3,000,000 tons per year is, of course, taking its toll directly from the operators, carriers and dealers alike. Thus, at present rates, well over 9,000,000 tons are moving annually over highways, which the various states conveniently keep clean of snow, and which are maintained at very low costs to the truckers. In many communities, yard dealers have been forced out of business through the competition of this form of distribution, and its sphere of operation is steadily growing. Trucked coal is being regularly delivered at points as remote from the mines as Rochester, Southern Vermont, Western Connecticut, Long Island and the District of Columbia.

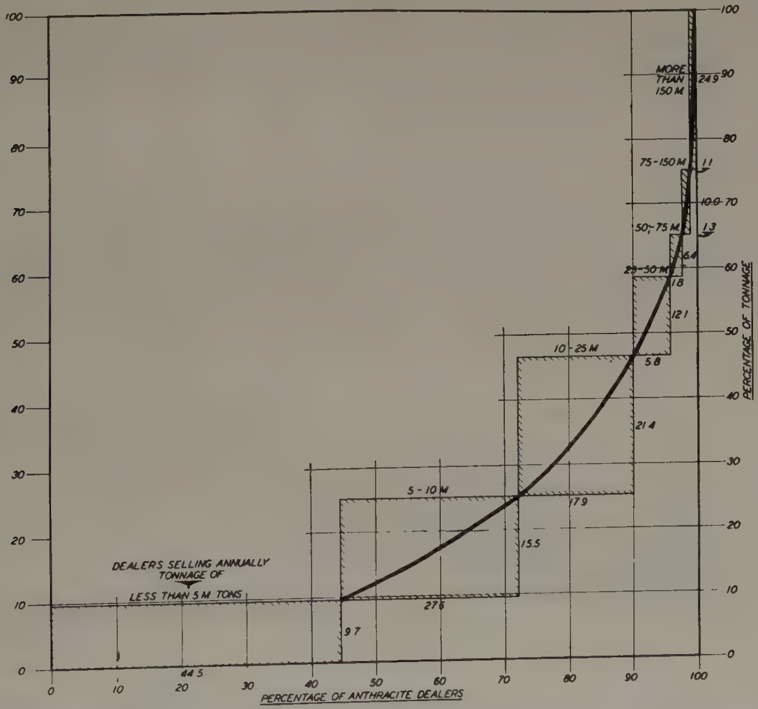

FIG. 2.—DEALER DISTRIBUTION OF ANTHRACITE SALES IN CONSUMING TERRITORY.

DEALER STRUCTURE

A recent survey covering some 2500 dealers of all sizes, located in about 600 cities and communities enclosed in a triangle bounded roughly by Quebec, the Twin Cities and Washington, disclosed that 44.5 per cent handled less than 5000 tons annually. What was more surprising, the sales of this 44.5 per cent constituted but 9.7 per cent of the total volume.

In Fig. 2, has been plotted the distribution of sales as between seven different sizes of dealers, the largest being those handling over 150,000 tons per annum. While these dealers accounted for but 1.1 per cent of the total number, their sales were about one-fourth of the aggregate. The different size classifications are indicated in the chart, together with their relation to the total number and to the total tonnage.

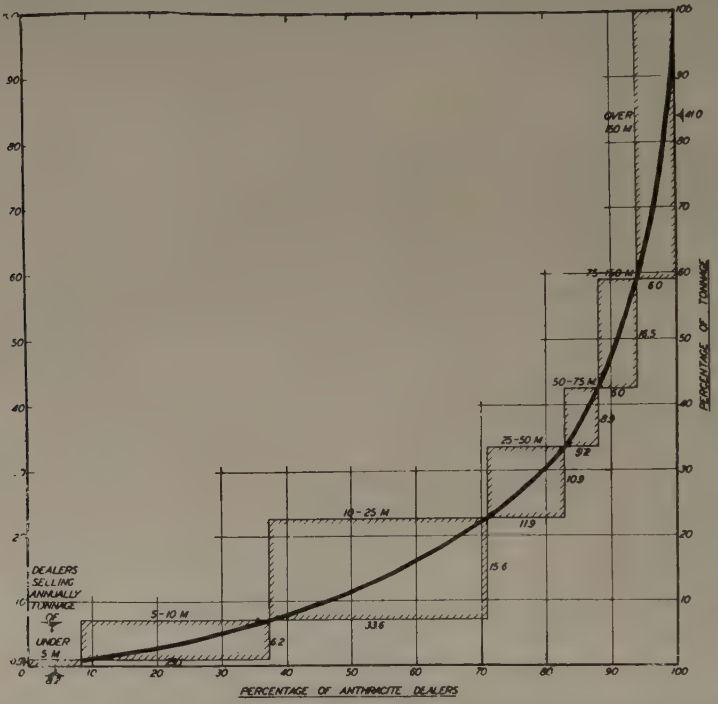

FIG. 3.—DEALER DISTRIBUTION OF ANTHRACITE SALES IN NEW YORK CITY.

One explanation for the unusually large proportion of small dealers with their relatively small quota of tonnage lies in the fact that many of them are in communities in which the total anthracite requirements are so small as to limit the number of dealers, but in which the competitive situation attracts a greater number than sound distribution economics would dictate. This duplication of facilities undoubtedly is responsible for a considerable portion of the cost of our distribution.

As a contrast to the chart, which is predicated upon the full survey, the data in Fig. 3 are plotted on the same basis, but only for dealers in the metropolitan area of New York City. The small dealers with less than 5000 tons per annum, instead of accounting for 44.5 per cent of the total number, constitute but 8.2 per cent, and instead of selling about 10 per cent of the total, contribute only 0.9 per cent. Thus is the curve widened at the base to indicate the squeezing out of the small and often uneconomical yards through pressure of the competition of the better merchandising methods, larger advertising expenditures, and more aggressive selling tactics of the progressive and better financed yards.

PRICES AND COSTS OF DISTRIBUTION

No paper on the economics of distribution would be complete without a word concerning the costs of distribution. But I doubt whether any such figure could be compiled for the anthracite industry as a whole. To determine it, we would have to consider the mine prices, which vary with the size and the season, the geographical destinations of each of the sizes, the freight rates for domestic, pea and steam sizes—which in many instances are different—the percentage shipped by rail, by rail and water and by truck, and then determine the dealers' margins, which vary not only with the size of anthracite, the size of the order, credit terms, carry-in, trimming, chute and other charges, but also with the month of the year. Also, there would have to be calculated large tonnages shipped direct to the consumer and the fact that a number of points in the West receive anthracite at less than the eastern circular, while at others, freight differentials are absorbed. The computation of an over-all figure of distribution costs is a study in itself.

However, the United States Bureau of Labor Statistics quotes each month what purport to be retail prices for chestnut size in several cities. Whether they are all on the same basis in the light of the many variations outlined above, is hard to ascertain. But taking those of Oct. 15, 1934, at which time the winter circular price was in effect, we find that the average for 25 cities was $13.17, without giving weight to relative distribution. The circular price for nut size, before allowance for cash discount, was $7. By a simple subtraction, the average freight rate and dealers' margin would appear to total $6.17, or 46.8 per cent of the average retail price. Lack of sufficient data precludes the possibility of a more exhaustive analysis of this figure, and it is merely offered for what it may be worth.

DISCUSSION

(*Cadwallader Evans, Jr., presiding*)

E. W. PARKER,* Primos, Pa. (written discussion).—In the anthracite industry the matter of distribution, as Mr. Patton has pointed out, presents some complicated and serious problems, almost, if not quite, as serious and complicated as those of mining and preparation. We really need a new type of engineer—engineers of distribution. A distinguished member of this Institute, a one-time president of the organization, and also, by the way, a one-time President of the United States, was, and I suppose still is, a profound student of the problems of distribution generally, and maintained that in the solution of them the recovery from the depression, which unfortunately began during his administration, would to a considerable degree depend.

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* Secretary-Treasurer, The Anthracite Institute.

I do not agree with Mr. Patton that the decline in the production and use of anthracite was primarily due to the invasion of other fuels. That invasion was a result—not a cause. The depression did not initiate the anthracite decline, though it accentuated and aggravated it. The decline in anthracite production began in 1925, four years before the financial crash, and continued for 10 years; last year, 1934, being the lone exception to that distressing decade. High labor costs and consequently high selling prices, together with the six months' strike in the winter of 1925–26, must accept the greater part of the responsibility for the conditions resulting in a loss of 40 per cent in the production of anthracite during that period.

The Anthracite Commission of 1920, then at the peak of the high cost of living, added a substantial increase (approximately 17 per cent) to the existing scale of wages. Not only has there been no recession in those rates, as there has been in practically every other line of industry, but when in 1923 another suspension of mining was threatened, the then Governor of Pennsylvania effected a "compromise" by laying that last straw in the way of a flat 10 per cent increase on all wage rates upon the already overburdened camel. Consumers naturally resented the increase they had to pay for their fuel and looked about for substitutes.

Then came the disastrous strike of 1925–26 in the middle of winter and substitutes became a necessity instead of a choice. So acute was the situation in New England that an appeal was made to the engineers of the Anthracite Service to teach New England householders how to burn bituminous coal in their furnaces built for anthracite. That appeal was relayed to headquarters in Philadelphia and the Service engineers were told to undertake this. It was felt that the anthracite industry should do what it could to keep the people of New England warm, if that were possible with bituminous coal. Gratitude has been defined as a lively appreciation of favors yet to come. What the people of New England may have expected in the way of additional favors we are unable to say, but certain it is that this helpful action on the part of the anthracite industry was rewarded by the loss of many thousands of tons in the New England markets, which, up to the present time, have not been regained, notwithstanding the fact that the anthracite-producing region is a highly profitable market for the products of New England factories. Also, New England imported at that time a small quantity of foreign anthracite. In 1927 it imported nearly 100,000 net tons. In 1933 it imported over 400,000 net tons and in 1934, over 450,000 net tons. How much of the continued use of bituminous coal is due to the difference in price between that produce and anthracite, and how much is due to the fear of another interruption to the anthracite supply (a string very actively played upon by the bituminous shippers), it is impossible to say.

It is apparent, however, that the citizens are willing to put up with the smoke and dirt of bituminous coal rather than to pay a trifle more for their customary and preferred fuel. The severe weather of last winter brought many of them back into the fold, and if this winter follows the example of last, a marked improvement will undoubtedly be shown in the anthracite business done in New England. The railroad shipments into eastern New York and New England as reported by the American Railway Association amounted to 99,753 cars in 1934 as compared with 83,389 cars in 1933.

The seasonal reduction in prices mentioned by Mr. Patton was inaugurated in the opening year of the present century, the purpose being to encourage more active buying by consumers during the summer months and thus distribute employment more evenly throughout the year, relieve the pressure on the storage yards and avoid the expense in handling and the loss by depreciation in storage of the surplus summer tonnage. As at that time prices of domestic sizes at the mines ranged from about

$2 to $3 per net ton and from $3.80 to $4 at New York tide (freight rates to tide were from $1.52 to $1.60 a net ton) the summer discounts (50¢ in April, 40¢ in May and so on until "circular" was restored in September) represented a substantial percentage to those taking advantage of them, and it was shown by the reports made to the United States Geological Survey that shipments of anthracite were fairly well equalized throughout the year.

With the advances in wages that began with the agreement of May 20, 1912, and which marked practically every conference until the "compromise" of 1923, and the coincidental advances in selling prices, the attractiveness of the summer discounts has lost its lure to such an extent that a dollar reduction at the present time does not encourage the early laying in of the winter supplies as much as 50¢ or even 30¢ did three decades ago, and again what to do with the summer production and equalize employment becomes a problem for the distribution engineer to solve.

Flow of Gas through Coal

By S. P. Burke,* Member A.I.M.E., and V. F. Parry†

(New York Meeting, February, 1935)

The presence of gas in coal mines necessitates the use of costly ventilation arrangements and the use of expensive mining methods. On the other hand, the gas itself in many instances is of considerable intrinsic value, and its separate recovery might conceivably be a profitable undertaking. This paper is concerned with a study of the movement of gas in coal strata with the object of determining the laws governing gas movement and thus making at least a preliminary step toward the solution of the more practical problem of its control or recovery.

Naturally, so important a subject has received much attention from scientists and engineers. In most cases, however, the studies have been concerned with the origin[1]‡, the amount[2], the composition[3] and the method of retention[4] of the gas in coal; or, from the more practical side, have been concerned with the phenomena and conditions associated with outbursts[5] or explosions and their control. Numerous studies also have been made of the permeability of coal to gas[6], of gas pressures measured in coal seams, etc. The authors are unaware, however, of any attempt to present a consistent or complete picture of the laws governing the movement of gas through the coal.

Study in West Virginia

A recent study conducted at this University[7] gives the first presentation of the distribution of pressures in a small section of a coal seam. Holes were drilled into the coal rib to various depths and at various distances from the coal face. The location selected for study was as remote as possible from disturbing influences of mining operations, faults, "wants," etc. The pipes extending into the coal that were used to determine pressures were grouted into the borehole for the entire depth of the hole, and great precaution was taken to insure accurate pressure readings. A map of the geology of the region, a map of the probable boundary and extent of the coal seam, a current map of the mine and a

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† Lecturer in Industrial Sciences, West Virginia University.

‡ References are at the end of the paper.

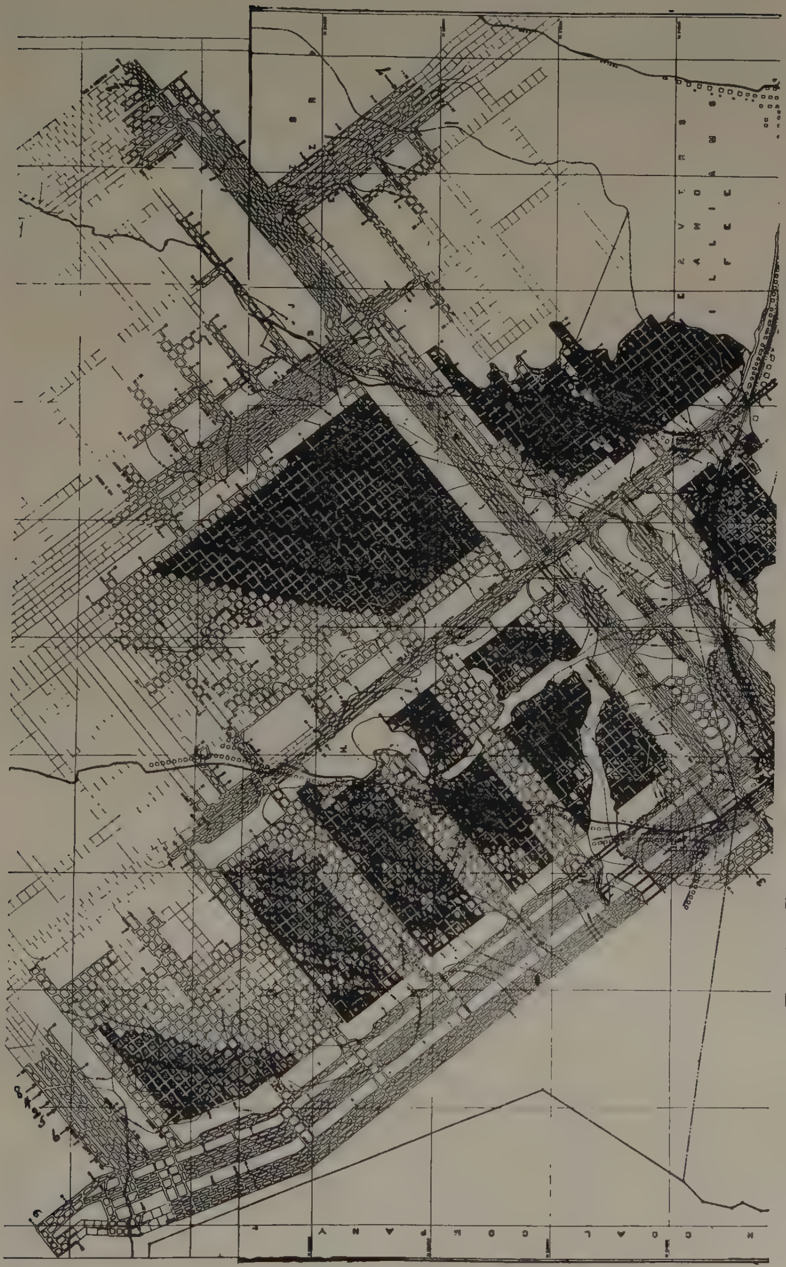

FIG. 1.—CURRENT MAP OF MINE WHERE GAS MEASUREMENTS WERE MADE. Drilled holes employed by Lawall and Morris are indicated by numbers.

detailed section of the location at which certain gas-flow measurements were obtained were given in the paper by Lawall and Morris[7]. The two last named are reproduced herewith as Figs. 1 and 2.

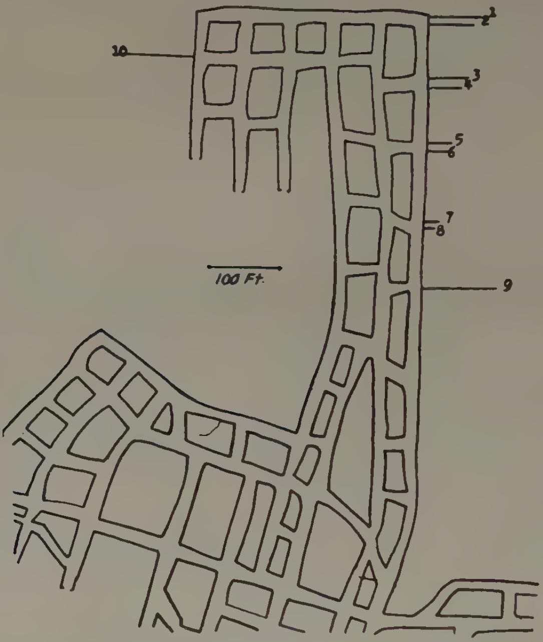

Fig. 2.—Detailed section of mine where gas measurements were made.

Table 1.—*Significant Data Used as Basis of Investigation*

Hole No.	Distance from Face, (*L*) Ft.	Depth in Coal (*D*), Ft.	Average Pressure as Read, Lb. per Sq. In. Gage	Absolute Pressure (*P*), Lb. per Sq. In.	P^2	$P^2 - Pa^2$	$\frac{P^2 - Pa^2}{D}$
1	3	70	17	30.95	957.7	763	10.90
2	15	60	12.9	26.85	720.7	526	8.77
3	89	50	7.0	20.95	438.8	244	4.88
4	97.5	40	6.7	20.65	426.7	232	5.80
5	175.5	30	2.6	16.55	280.7	86	2.87
6	186.5	25	2.75	16.75	273.9	79.2	3.17
7	272.0	20	1.0	14.95	223.5	28.8	1.44
8	285.0	15	0.5	14.95	208.9	14.2	0.946
9	363.0	102	5.5	19.45	378.3	184	1.80
10	60	95	9.5	23.45	550.0	355	3.74

Pa = average barometric pressure = 13.95 lb. per sq. in.

The significant data employed by the present authors are reproduced in the first four columns of Table 1. (The average pressures given are estimated from the data presented by Lawall and Morris and differ slightly in some cases from their estimates.) In attempting to interpret these data, it was found desirable to continue the study by a determination of the rate of gas liberation from the surface of the coal rib at locations directly behind which the pressures had been determined. Accordingly, apparatus was constructed to obtain these measurements, with which Mr. Bell, of the Consolidation Coal Co., obtained the data given in Table 2. The measurements were made in positions immediately adjacent to the drilled holes employed by Lawall and Morris[7] in determining gas pressures (upper left corner of Fig. 1).

TABLE 2.—*Rate of Gas Liberation*

DISTANCE FROM COAL FACE (*L*), FT.	VELOCITY OF FLOW, v CU. FT. PER HR. PER SQ. FT
22	6.32
104	1.74
193	1.86
289	1.0
365	1.1

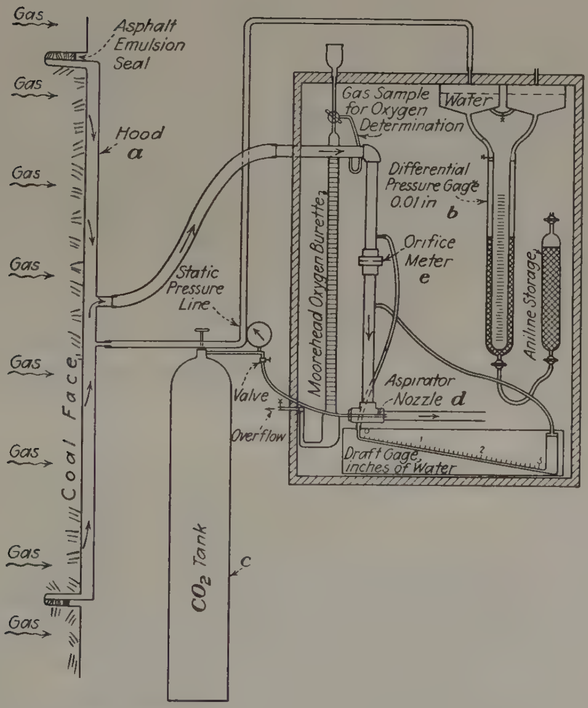

FIG. 3.—EQUIPMENT FOR MEASURING FLOW OF GAS FROM COAL FACE.

APPARATUS

The apparatus employed for the determination of the rate of gas liberation is shown schematically in Fig. 3. The surface of the coal at the place where a test was to be run was made as smooth as possible. A groove an inch or more wide was then cut in the coal into which the air-tight sheet-metal chamber *a*, approximately 4 ft. on a side, was fitted. The sides of the chamber were caulked or mudded into the groove so that there would be no leakage of gas to or from the chamber around the edges where contact was made with the coal. As a result, an area of approximately 16 sq. ft. of coal surface was covered by a gastight metal box. By means of a differential manometer *b*, the pressure difference between the inside of the chamber and the outside atmosphere could be determined to approximately 1⁄1000 in. of water. A cylinder of carbon dioxide *c*, together with a nozzle *d*, was employed as an injection suction pump by means of which gas could be drawn from the chamber at any desired rate. A carefully calibrated orifice meter *e* was used to determine the rate at which gas was being drawn from the chamber.

The procedure in making a determination was as follows: After the metal box had been caulked in position on the coal surface, and the system had been tested for gastightness so far as possible, the carbon dioxide was turned on and gas removed from the chamber by injection. The rate of flow of carbon dioxide was carefully adjusted, so that the differential manometer indicated that the pressure inside the chamber was exactly equal to that of the outside atmosphere. Under these conditions, the rate of gas flow as indicated by the orifice meter was recorded. Obviously, a great number of observations could be taken rapidly. At each position, a series of determinations was made over the interval of a few days' time. The data recorded in Table 2 are the averages of the results obtained.

It is important to emphasize that, during the period when the data reported in Table 2 were obtained, simultaneous measurements of the pressures in the adjacent drilled holes showed no appreciable change from the averages reported by Lawall and Morris[7].

Interpretation of Data

In attempting to arrive at any consistent theory of the mechanism of gas movement through the coal, it is convenient to consider various hypotheses concerning the source or distribution of gas in the coal bed. For example, we may consider that the gas is stored in one or a finite number of reservoirs situated a finite or an infinite distance from the point where the gas is being liberated. In this case, the coal itself simply becomes a permeable medium through which the gas moves. On the other hand, it may be assumed that the gas is associated with the coal itself, existing either in interstices under pressure, or in actual solution or sorption in the coal, in which case the coal bed itself serves as the gas reservoir. In either case, the known adsorption of gas by the coal is a complicating factor.

Now, regardless of what hypothesis is made, it is extremely important for purposes of our analysis to know whether or not the conditions being investigated correspond to a case of "steady" movement of the gas. Fortunately, the evidence on this point appears definite. The driving of the mine entry in which this study was made was completed by July 1, 1931. The investigation of pressures reported by Lawall and Morris was begun in September, 1931. No mine operations were carried on in the entry after its completion in July and the nearest operation was carried on in the perpendicular entry over the period from September, 1931, to the completion of the investigation reported here, in July, 1932.

Thus, over the period of more than nine months, the average pressures in the various drill holes remained substantially constant. We can assume, therefore, either that we are dealing with a case of steady flow, or at least that the flow may be considered steady over the area of the field investigated and during the period of time considered.

The equation of continuity for the steady flow of a compressible fluid in two dimensions is as follows:

$$\frac{\partial(\rho u)}{\partial x} + \frac{\partial(\rho v)}{\partial y} = 0 \qquad [1]$$

where u and v are components of the velocity in the x and y directions respectively and ρ is the density of the fluid. Let us further make the assumption that the gas actually flows through interstices, cracks or pores in the coal seam and that its movement obeys the law of flow of gas passing through a porous, homogeneous, isotropic solid. This law may be represented as follows[8]:

$$\frac{dp}{ds} = -\frac{k\rho U^2}{\delta}\left(\frac{\mu}{\rho U \delta}\right)^{2-n} \qquad [2]$$

where k is a dimensionless constant, U equals the velocity in the s direction, μ equals the viscosity of the fluid, δ is a shape factor of dimension equal to length. If we assume further that the unabsorbed gas obeys Boyle's law, that conditions are isothermal, and that n is constant, the foregoing equation reduces to the following:

$$-K\frac{dp^2}{ds} = (\rho U)^n \qquad [3]$$

where K is a function of the geometrical character of the pores, of the molecular weight of the gas, of the temperature, of the viscosity and of the nature of the flow n. Resolving equation 3 into its components gives:

$$\frac{-K\dfrac{\partial p^2}{\partial x}}{\left(\dfrac{\partial p^2}{\partial s}\right)^{\frac{n-1}{n}}} = \rho u \text{ and } \frac{-K\dfrac{\partial p^2}{\partial y}}{\left(\dfrac{\partial p^2}{\partial s}\right)^{\frac{n-1}{n}}} = \rho v \qquad [4]$$

where
$$\frac{\partial p^2}{\partial s} = \left[\left(\frac{\partial p^2}{\partial x}\right)^2 + \left(\frac{\partial p^2}{\partial y}\right)^2\right]^{1/2}$$

At the surface of the coal rib, where the flows were measured, $dp^2/dx = 0$, since the pressure in the entry was always constant and equal to approximately one atmosphere. Fig. 2 shows the positive directions for the ordinates x and y. Hence, at the surface of the coal, the component of flow in the x direction is equal to 0 and equations 4 reduce to the expression

$$-K\left(\frac{\partial p^2}{\partial y}\right)^{\frac{1}{n}} = \rho v$$

which, since ρ is constant at the coal surface, becomes

$$-K'\frac{\partial p^2}{\partial y} = v^n \qquad [5]$$

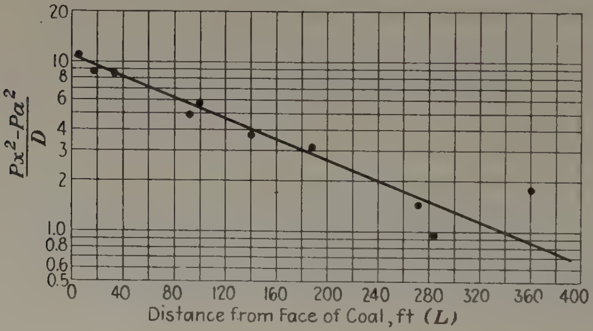

FIG. 4.—PRESSURE DATA PLOTTED ON SEMILOGARITHMIC PAPER.

Obviously, if the expression for the pressure in terms of x and y were known, it could be substituted in equation 5. Moreover, since the velocity of the flow of gas from the face of the rib has been determined experimentally, it would then become possible to determine the value of n and K' in that equation.

The pressure data presented in Table 1 are shown in Fig. 4, plotted on semilogarithmic paper. The quotient of the difference of the square of the pressure and the square of the atmospheric pressure divided by the depth of the drill hole is plotted as the logarithmic ordinate, while the distance measured from the end of the entry (that is, the face of the coal) is plotted arithmetically. The results for the most part show fair agreement with the straight line drawn on the graph, bearing in mind that the coal is not uniform and that inaccuracies due to slight leaks, etc., must inevitably occur. The equation of the straight line shown in the graph is as follows:

$$\log \frac{P_x^2 - P_a^2}{D} = -0.00308L + 1.033 \qquad [6]$$

There is no special merit in this method of mathematical approximation. Other types of equations would fit equally well. However, this equation leads to a reasonably good mathematical statement of the data. Fig. 5 shows the isobars in the coal seam computed from formula 6. For comparison, the actual data are plotted on the same figure, and there again it is seen that the agreement is fairly good. In Fig. 6, the solid portions of the curves represent the mathematical isobars as plotted in Fig. 5 and the extrapolated dotted curves show the extension of the isobars as they probably exist in the coal. It is evident that the formula of equation 6 is a good fit only over the region of our actual investigation. From equation 6, we have the following

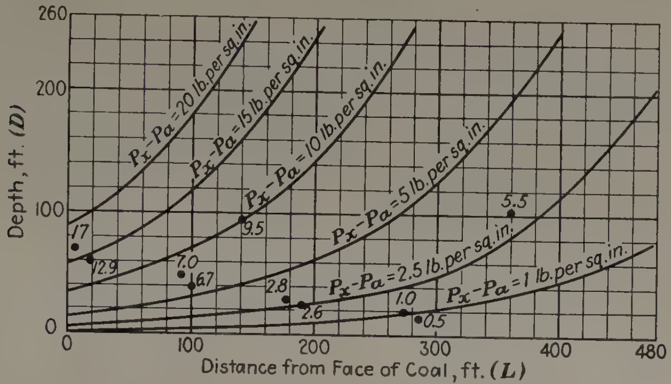

FIG. 5.—RELATION BETWEEN PRESSURE AND DEPTH. ISOBARS IN COAL SEAM COMPUTED FROM FORMULA 6.

$$P_x^2 - P_a^2 = 10^{(-0.00308L+1.033)} \cdot D$$
$$= -y \cdot 10^{(-0.00308L+1.033)}$$

differentiating

$$\frac{dp^2}{dy} = -10^{(-0.00308L+1.033)} = \frac{-v^n}{K'}$$

Let

$$(-0.00308L + 1.033) = X$$

then

$$\log v = \frac{X}{n} + \frac{1}{n} \log K' \qquad [7]$$

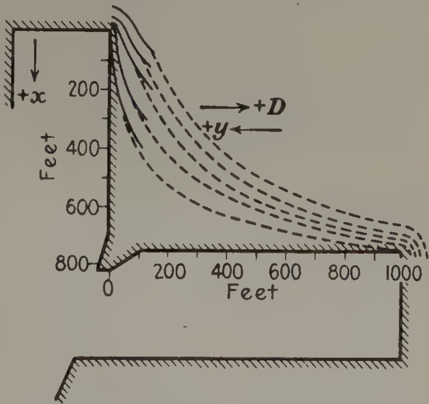

FIG. 6.—SOLID LINES SHOW ISOBARS AS PLOTTED IN FIG. 3; DOTTED LINES, PROBABLE EXTENSION OF ISOBARS.

which expresses the relation between the velocity of gas flow at the surface of the coal in terms of the physical properties of the coal and of the nature of the gas flow—whether turbulent or viscous—on the assumption that the law of flow is as stated in equation 2. To evaluate equation 7, the flow data obtained experimentally and reported in Table 2 were plotted semilogarithmically, as shown in Fig. 7. The solid line is the best straight line curve that can be drawn through the plotted points. The equation of this line is

$$\log v = -0.00308L + 1.033 - \log 1.463 \qquad [8]$$

Unfortunately, time and facilities did not permit additional flow data to be obtained. Conclusions cannot be drawn from Fig. 7 with much confidence. Many additional data are needed, but as there appears little hope that these can be obtained in the near future, the authors feel justified in presenting the available data at this time. In any event, equation 8 appears to be the best representation of the few data that are available. Moreover, it is interesting that the slope of the line is such that the factor n in equation 7 is almost exactly equal to one, thus indicating that the gas goes through the coal in viscous or streamline flow. This is in accord with the evidence of the flow of natural gas through gas sands, which indicates that in those geological strata the movement of the gas is also viscous.

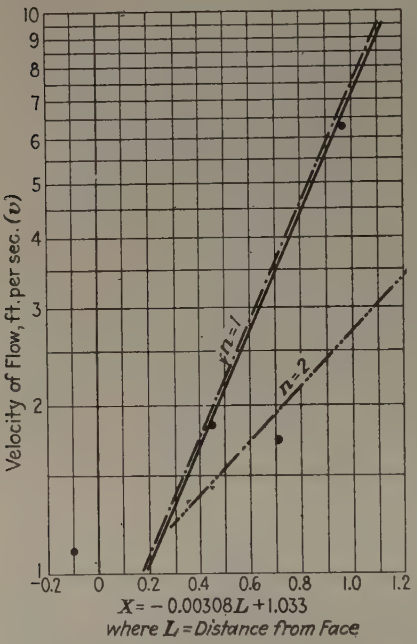

FIG. 7.—FLOW DATA OF TABLE 2.

Permeability of Coal in Situ

From the value of the factor K' in equation 7 ($K' = 1/1.463$), it is possible to evaluate the permeability of the coal seam in situ. From equation 5, since $n = 1$,

$$\frac{\partial p^2}{\partial y} = -\frac{v}{K'}$$

or

$$2p\frac{\partial p}{\partial y} = -1.463v$$

Since the pressure at the surface of the coal was maintained at approximately 13.95 lb. per sq. in.,

$$\frac{\partial p}{\partial y} = -0.0524v$$

Converting from English to metric units (dyne cm. sec.)

$$\frac{\partial p}{\partial y} = -14{,}000v$$

Hence, in equation 2

$$\frac{k}{\delta^2}\mu = 14{,}000; \text{ or } \frac{k}{\delta^2} = 1.27 \times 10^8, \text{ since } \mu \text{ for } CH_4 = 1.1 \times 10^{-4}$$

$$\therefore \frac{\partial p}{\partial y} = 1.27 \times 10^8 \mu v \text{ (c.g.s. system)}$$

For water, $\mu = 0.01$. If $\frac{\partial p}{\partial y} = \frac{1 \text{ gram}}{\text{sq. cm.}} = 981$ dynes per sq. cm., then $v = 77{,}000 \times 10^{-8}$. This value for the permeability is approximately equal to that experimentally determined for the more permeable gas sands of Pennsylvania[9].

Gas Entering Mine

If, as has been assumed, the coal is macroscopically homogeneously porous in all directions, an estimate of the amount of gas entering the mine from the block of coal shown in Fig. 6 can be obtained by integrating equation 8 over the entire length of the block (800 ft.), or by extrapolating the data shown in Table 2 over the same area. The latter is the preferable method, since equation 8 is really applicable only to the area actually investigated. By extrapolating the data of Table 2 and integrating to determine the total flow from the face of the coal and projecting the area of the coal rib on to the general perimeter of the mine, it is found that approximately 250,000 cu. ft. of methane enters the mine per 1000 ft. of perimeter per 24 hr. Since the over-all perimeter of the mine is approximately 26,000 ft., this computation indicates that approximately 6½ million cubic feet of gas enters the mine per 24 hr. It must be remembered, however, that the data in Table 2 were measured at one corner of the mine and, moreover, part of the mine perimeter is made up of "wants" from which it is expected that little or no gas would flow. Taking these factors into account, one may assume that the actual flow of gas would not exceed two-thirds of the figure just computed.

In Fig. 8 are shown, through the courtesy of the Consolidation Coal Co., charts indicating the flow of gas from the mine since the mine was opened in 1924. At the time these experiments were conducted, when the actual coal production was very low, the amount of methane liberated was approximately 4,200,000 cu. ft. per 24 hr., a figure that is in reasonable agreement with that just computed.

Fig. 6 shows that when the average coal production is 2000 tons per day (1931), the average methane production is approximately five and three-quarter million cubic feet per day, or an increase of approximately one and one-half million cubic feet of gas accompanying an increased production of 1400 tons per day.

This apparent relation between the production of coal and gas evolution has led many observers to assume that the gas evolution is primarily due to the breaking up of the coal as it is mined. A brief consideration will show the fallacy of this conclusion. The relation between pressure and depth shown in Fig. 5 indicates that coal as it is mined will rarely be broken from a region from which the gas pressure is greater than 15 lb. per sq. in. gage. Let us assume that 2000 tons of coal in equilibrium with gas at two atmospheres absolute pressure is broken down from the seam. The data of many observers[4] show that the gas content of coal of this type at this pressure is approximately twice the volume of the coal. In other words, assuming that 2000 tons of coal were completely to yield up its gas, the total volume of gas produced would be approximately 400,000 cu. ft. This, of course, will by no means account for the known yield of five and three-quarter million cubic feet, and does not even account for the one and one-half million cubic feet that is produced as a result of the increase of coal production cited above, since more than one million cubic feet is left unaccounted for. The production of the latter, however, is readily understood, since advancing the headings into the areas of higher gas pressure will greatly increase the rate of production of gas from the surface of the coal ribs in these galleries or entries.

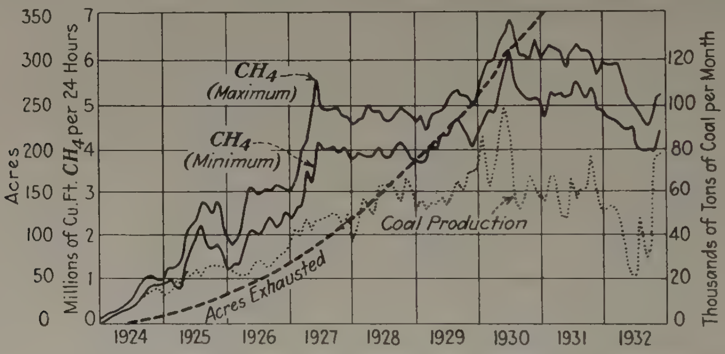

FIG. 8.—AMOUNT OF METHANE PRODUCTION IN RELATION TO COAL PRODUCTION.

SOURCE OF THE GAS

Much has been written regarding the source of the gas associated with the coal in this and similar seams, but as yet little evidence is available upon which conclusions can be based. Unfortunately, our study does not permit any precise deductions to be drawn on this phase of the subject, but certain interesting observations can be made.

According to the best geological information we can obtain, the known area of this particular coal stratum is approximately 150 square miles. The maximum thickness of the seam is approximately 5 ft. Much of the seam is appreciably thinner than this. Assuming, however, a prevailing thickness of 5 ft., the total volume of the seam is approximately 21,000,000,000 cu. ft. A study of the topography indicates that the hydrostatic head of the seam, that is, the distance of the coal below stream level, is about 500 ft., thus giving a hydrostatic pressure on the coal of about 230 lb. per sq. in., or about 15 atmospheres. It is significant that water flows through the coal seam at several points in this mine. Referring again to the experiments that have been conducted on the sorption of methane in coals of this type[4], we find that at this pressure the maximum gas that should be associated with the coal is approximately six times its volume. Thus, the total gas associated with this particular coal stratum should be approximately 126,000,000,000 cubic feet.

Integrating the curve of Fig. 8 that shows the production of methane indicates that since the mine was opened more than 13,000,000,000 cu. ft. have been released. In addition, several other mines in this region are known to be equally "gassy" and to have been in operation for at least the same length of time. Moreover, a large section of the seam outcrops, and this outcropping has so drained the coal of gas in the eastern region that drift mines or shallow shaft mines in this location are "non-gassy." Therefore, we can readily account for a volume of gas already released from this coal seam which is at least as great as the amount of gas we have computed as being associated with the coal. On the other hand, the gas pressures in the coal seam in the region of this mine remain constant over a period of at least nine months. Moreover, we find no evidence of diminution in gas production from this mine during the latter years of its operation.

Consequently, we are forced to conclude: (1) that the gas really exists, in at least a part of the coal seam, at a pressure much greater than the hydrostatic head; or (2) that the "solubility" of gas in this coal is very much greater than that of any other similar coal that has yet been measured; or (3) that the coal is at present undergoing metamorphosis and is generating gas at this rapid rate; or (4) that the real source of the gas is from some reservoir beyond the confines of the coal seam.

Of these hypotheses, the last two seem the more tenable. Yet it has been observed that, while generally speaking, "coal" gas is found associated with coals such as this, of high fixed carbon content, geologists tell us that the occurrence of natural gas in commercial quantity is not to be expected in regions where such coals exist[10]. Several gas wells have been drilled in this territory, but no commercial quantities have been recovered.

Accordingly, then, it appears that we may have evidence at hand that this coal is undergoing metamorphosis at such a rate that in a period of 10 years (assuming a gas production in this time of two hundred billion cubic feet) it has lost 0.4 per cent of its mass. This suggestion, while highly improbable, is interesting nevertheless.

SUMMARY

This paper presents a new method of approach to a study of the phenomena of the movement of gas in coal seams.

Definite laws of flow governing the movement of gas in coal seams are presented, based upon the available data.

As a corollary, it is shown that the movement of gas in coal is viscous in character.

The permeability of coal in situ for a depth of 100 ft. in the coal rib has been determined and is shown to be apparently equal in Pocahontas No. 4 coal to the more permeable gas sands in the eastern fields.

The mechanism of the production of gas by a gassy coal mine is described.

Certain interesting observations and various deductions therefrom concerning the source of gas in coal are presented, which indicate the possibility that the coal studied *may* be undergoing metamorphosis at a surprisingly rapid rate.

Acknowledgment

The authors are particularly grateful to Mr. Thomas G. Fear, formerly of Consolidation Coal Co., and to the company itself, for their courtesy and assistance in prosecuting this investigation. In particular, also, acknowledgment is made to Mr. Homer Bell, of the same company, for his aid in obtaining certain of the data. The authors also acknowledge their indebtedness to the prior investigations of Prof. C. E. Lawall and Mr. Lee Morris, upon which this study is largely based.

The present study is entirely preliminary in nature and is presented because its continuation in the same mine at the present time is practically impossible and in the hope that the coal industry may cooperate in advancing this investigation, which has as its basic objective the control of a terrible coal-mining hazard.

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DISCUSSION

(*E. A. Holbrook presiding*)

G. S. Rice,* Washington, D.C. (written discussion).—The device that was employed in the Consolidation Coal Company's mine in the Pocahontas coal field of West Virginia, to determine the flow of gas through a unit area of surface of the coal face, is most ingenious and should give accurate results for that area, but does so small a unit area as 4 ft. square give values for gas flow that can be applied to large areas in most coal strata? It would more nearly do so in the Pocahontas No. 4 coal bed than any other coal stratum with which I am familiar, but in most coals firedamp or gas comes out irregularly in different parts of the mine. It may come from different layers in the coal seam or from roof or floor and is materially affected by rolls, faults and clay veins.

It appears to me that a more accurate method for a large area or for a whole mine is to determine the amount of gas given off in different ventilating splits of the mine and employing precise methods of sampling and analysis over considerable periods of time. It is well known that the volume of gas given off varies widely as a mine advances or stops, or special geological conditions are encountered, and the period must be long enough to insure true averages.

It is admitted that it is difficult to relate the area of face exposed to each particular ventilating split, but it is even more difficult to do that employing a unit area of surface only 4 ft. square.

The authors' discussion of the possible origin of the enormous flow of firedamp into the particular group of Pocahontas mines is interesting. Many of us have studied this question but, like the authors, find there is no simple answer—it remains a continuing problem.

G. Roblings, Ystradgynlais, South Wales (written discussion).—This matter of the flow of gas in coal is still somewhat obscure, inasmuch as conditions existing in the mines have so many variables that it becomes difficult to arrive at a conclusion that satisfactorily answers the question when applied to such a multitudinous variety.

These variations apply to the character of the coals over even limited areas, roof and floor, interstratified layers of impurities such as clod or rashing; proximity, extent and nature of disturbances. Since any or all of these have some effect upon the release of gas from coal seams, testing by an instrument, however ingenious, covering a few square feet can only be of academic interest.

It is open to question whether the gas that the authors have trapped has really flowed out or passed out through the coal. Dr. Harger[11] stated that methane was held in coal in two physical conditions; one part was given off rapidly as the coal was being worked and the adsorbed part was quite distinct and could be held by the coal for many years. If, however, the coal was full of breaks or with cleavage highly developed, the first-mentioned portion would be expected to get off very rapidly, but in other more compact seams of coal the amount of gas released is very small and it has been found that there is no relationship between it and that retained in the adsorbed state.

The writer has had considerable experience with outbursts at a colliery in the anthracite portion of the South Wales coal field and from his observations he cannot accept that there is any substantial flow of gas through the coal in situ even when this is so friable that it can be released by scratching with a pick point. In one such patch where hammering with a mallet would have brought about a serious outburst owing to the nature of gas content (which was being crossed by a heading 10 ft. wide, the coal being 6 ft. thick), in which in an area of 60 sq. ft. nine boreholes each 3 in. in diameter were bored to a depth of 9 ft., substantial quantities of gas (methane, 98.8 per cent, Seyler) were released with the bored-out coal, yet the gas in the coal immediately surrounding the holes remained in the same dangerous state in which it existed prior to boring. In fact the cessation of boring in any hole in any position was followed by an immediate cessation in the release of gas. In other patches it was subsequently found that even after considerable intervals of time the coal had retained this gas and proved it by bursting out with dangerous quantities of coal and gas.

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* Chief Mining Engineer, U.S. Bureau of Mines.

[11] Harger: Fed. Int. Min. Engrs., **45**.

After several outbursts induced by heavy blasting, when the coal was cleared it was found that the adjoining coal was still dangerous to work and further outbursts were induced, indicating that the coal had retained its gas although crushed to an amorphous mass of small slickensided scales.

In another case, it was found that after three years gas had escaped from a pillar adjoining an outburst area and had been replaced by moisture due to water having gathered at the rise side of the pillar and percolated it; the coal thus in its greater avidity for moisture had released the gas. This was found when an outburst was induced in the coal, the effect of which extended up to within 9 ft. of the above area, and on cutting through the coal in the pillar was found to be quite damp.

In a colliery on the Continent visited by the author, in which outbursts had been frequent and troublesome, a considerable quantity of gas was being continually given out by the coal, and to such an extent as to make efficient ventilation difficult. There was a condition totally unlike that which the writer had experienced.

The writer has noticed in a seam with an interstratified rash that a flame safety lamp could not be kept upon the rash because of the gas released from the rash, whereas no such phenomenon was noticeable in the coal itself.

Also, trouble can be caused in other seams by gas from the roof, and the writer has experienced in one seam more trouble from this cause than any from the coal seam itself.

These tend to show that little value can be attached to investigation as to release of gas from coal in itself, which may not be more than the gradual loss of power to retain it in adsorption owing possibly to the greater avidity of the coal for the moisture in the air.

What is of far greater importance is the total amount released by the coal and the surrounding measures together, and its effect upon the ventilation of the mine.

Conserving Health of Employees in the Coal Industry

By Fred A. Krafft*

(New York Meeting, February, 1936)

The time allotted to this paper will permit only the sketching of general principles and practice as generally employed in the industry to maintain and preserve the physical well-being of the miner and his family. Let it be emphasized on behalf of the industry that coal operators, individually and collectively, have given primary consideration to the appointment of a mine or community physician who is charged with the responsibility of caring for the sick and injured. However, it too frequently happens that with the selection or appointment of the camp doctor many operators wash their hands of further responsibility toward the health and the working efficiency of their employees.

The coal industry is so organized in most of the coal-producing areas that not only is the health of the individual miner an important production problem but the collective health of the entire mining community is a fundamental operating problem. This is particularly demonstrated when epidemics of communicable diseases attack entire communities, with resultant loss of life and production and general community distress.

The problem and responsibility of maintaining high health standards among miners and their families should be given as much consideration as safety, economical production and efficient distribution are given in the successful operation of coal mines.

Company Policy

In the inauguration or administration of a medical program for a large mining operation or for an individual mine, the first question to be solved is what the company policy is to be with respect to the following factors pertinent to a medical program:

1. On what basis will the company and the employees finance the program?
2. Should a contract doctor or a full-time doctor on salary be employed?
3. What measure of service is to be rendered the employee and his family in return for his monthly contribution to the medical program?

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4. What standards of housing and sanitation will the company maintain?

5. What program of public health service and education will the company maintain?

6. What arrangements or provisions will be made for the hospitalization of illness and injury that cannot be properly treated in the home or doctor's office?

1. The direct operating costs of a medical program should be supported entirely by the income received from the employees in the form of a monthly medical check-off. Where this condition or arrangement prevails, the employees, individually and collectively, are inclined to be exacting; they will expect the best possible service because they are paying for it. The employees and their families will be more cooperative in matters of health education and community sanitation because they will feel a proprietary partnership in the program. If the employee pays $12 to $18 per year for himself and family he is buying health insurance and health assurance without overburdening his pocketbook, and as these are purchased on a mass production basis they save him, if he or his family have any accidents or sickness, possibly $100 or more per year in doctor and drug bills. And finally, the company contribution to the program should include the expense of checking and collecting each month the medical dues.

The administrative costs to the company should not exceed 5 per cent of the medical check-off, and this expense will be more than returned to the company in reduced compensation costs, reduced absenteeism, improved physical condition and increased production efficiency of the miner and improved sanitary conditions, thereby reducing to a minimum all of the factors that enter into the production costs of coal mining.

2. Should a contract or full-time company doctor be employed? Circumstances and conditions may affect the decision. In our experience with the two different set-ups we strongly favor the full-time company physician, and we know that our employees have the same opinion. This is intended as no reflection on contract doctors, many of whom are as conscientious and as thoroughly competent as the full-time company physician. But the company organized and supervised medical program and the full-time salaried company physician have demonstrated their efficiency and benefits.

3. What measure of service is to be rendered the employee and his family in return for his monthly medical check-off? Conditions and circumstances affect opinions or decisions. In the main the employee and his family are entitled, in consideration of his medical dues, to free medical service and free drugs. The doctor should be provided with a modern and sanitary office with modern facilities and equipment enabling him to practice all phases of modern medicine. All applicants for employ-

ment and the old employees and their families should be immunized against typhoid and scarlet fever, smallpox and diphtheria at no cost to themselves.

A well rounded program should provide for periodical analysis of the milk and water supply, and the necessary precautions should be taken to assure a milk and water supply that meets with modern public health standards and requirements.

In an extensive mining operation where stores, recreation buildings, clubhouses, restaurants or similar enterprises are a part of the community, an examination of foodhandlers should be made at least twice a year at no cost to those affected. Where communicable diseases are discovered a diligent follow-up for the correction and cure of such disease should be a part of the program.

Physical examinations for all new employees and periodical examination of all old employees should likewise constitute an important phase of the program. The medical program and personnel should be coordinated effectively with the production department, or any other departments responsible for the general operations of any mining community, and every effort should be made by the medical department and the other departments to make each and every employee public-health-conscious.

4. In any mining operation involving a community set-up wherein an extensive house plant is an adjunct of the mine operations, provision should be made for the erection and maintenance of sanitary toilets and the periodical sanitating of such toilets; garbage cans should be provided for each householder at no cost to the employee except a deposit to assure the return of the garbage can to the company, and weekly disposal of garbage should be maintained by the company at a nominal cost to the employee.

We have found that periodic sanitary inspections have materially reduced the cost of garbage disposal, house-plant maintenance, and in general have greatly improved the community appearance of our house plant and raised the general level of health within the community.

It goes without saying that residents of mining communities cannot be kept in general good health if the house plant itself is not properly maintained. Leaky roofs and side walls or houses located in low or damp land are not conducive to good health. Congested tenancy—for instance, 10 human beings housed in a four-room shack—will tax the ingenuity and ability of the most proficient doctor.

5. What program of public health service and education will a company maintain? In our organization, embracing an average working force of 10,000 to 12,000 and community populations ranging from 1,000 to 12,000, we have found it beneficial to the company, its employees and the communities at large to maintain, in conjunction with our medical program and staff, a group of public health nurses. All of our nurses

have had basic institutional training and experience, augmented by public health training in recognized schools.

In addition to assisting the company physician during office hours and following up the more serious medical cases in the homes, our nurses conduct a public health program in the homes of the various communities, particularly in the schools. This program, spotting only its highlights, involves the following features:

a. Sanitation.
b. Preparation of foods.
c. Care of babies.
d. Diet and nutrition.
e. Instruction in bedside care.
f. Observation of symptoms.
g. Temperature and respiration.
h. Applying cold, heat and counter irritants.
i. Care of communicable diseases.
j. First aid of common ailments and emergencies.
k. Clinics:
    (1) Prenatal.
    (2) Infant.
    (3) Dental.
    (4) Tuberculosis.
    (5) Nose-throat.
    (6) Orthopedic.
    (7) Immunization.
l. School health examinations.

We have found it extremely beneficial to all interests concerned to organize Home Hygiene classes and Care for the Sick classes among the older women and young women of each community, so that we are prepared to meet such emergencies as a mine catastrophe or an epidemic such as influenza, typhoid fever, smallpox and diphtheria.

We have found that our nurses are employed to advantage in assisting our doctors in our immunization program, which involves the miner and every member of his family. During the past several years, in cycles of three years each, we have immunized against typhoid, scarlet fever, smallpox and diphtheria practically 95 per cent of our adult population as well as our children. The immunization of the children is made possible and comparatively simple through cooperation with the local school systems.

Prior to the inauguration of our immunization program in one of our large Kentucky divisions, it was not uncommon, in a population of approximately 10,000, to treat 200 to 225 cases of typhoid fever each year, with a resulting death rate of 10 to 12 per cent. For the past seven years, as a result of our immunization program, we have treated only 14 cases of typhoid, mostly transitory, and among these 14 cases there have been two deaths.

Anyone operating a coal mine, or a group of coal mines, in communities of small population, appreciates not only the human suffering and loss of life, but the economic loss that is occasioned by an epidemic that closes down mines and schools. A public health program if it eliminates such distressing and costly experience is in itself a paying investment; however, there are larger returns than we have mentioned. The growing of gardens and the canning of foodstuffs, instruction in the preparation of foods, teaching our employees how and where to get the most for their money in food values, are only a few of the educational features that can play an important part in a public health program.

It has been said, and rightly so, by many industrial leaders familiar with these things that a community doctor and a community nurse can and do serve their employers as emissaries of good will for the company in contacting not only the male workers of the organization but members of their families. To a certain degree these professional people can be made to serve their employers in a capacity that involves all phases of human relations and employee relationships. A good mine superintendent or a good manager of any operation will maintain a close contact with the company doctor and the company nurse, as a matter of good business, and through them maintain closer contacts with his employees.

When a company is organized in units large enough to justify the employment of a full-time salaried doctor and public health nurse, it is essential that the activities developed and administered by the medical section of the department should head up under the supervision of one company official, especially qualified to coordinate the activities of these two branches of the program.

There can be no room in a program such as we have described for petty professional jealousy or so-called professional ethical barriers existing between the doctor and the nurse. The service that each renders to the community, the company, and to the employee and his family, must be predicated on the theory that their joint responsibility is to keep the miner and his family well. Personal interests and self glory cannot take precedence over service and care of the miner and his family.

6. What arrangements or provisions will be made for hospitalization of illness and injury cases that cannot be properly treated in the home or in the doctor's office? There has been a general movement on foot in this country for the past several years to recognize the need for a combined hospitalization and medical program that will serve industrial workers and their families on the so-called insurance basis.

The most distressing experience in a miner's life involves that wherein he or a member of his family finds it necessary to seek hospital attention and the service of a surgeon. Invariably such service and operation costs anywhere from $100 to $200, this amount deducted from the annual earnings of coal miners in nine out of ten cases works a hardship which is not easily overcome and as a matter of fact hangs as a cloud over the miner's head indefinitely.

Those who are familiar with hospitalization programs in certain portions of West Virginia, Pennsylvania and Kentucky are aware of the fact that a miner can insure for himself and his family against such an experience for a monthly contribution ranging from $2.50 to $3.00 per month. Hospital statistics show that hospitals that provide this type of service are patronized at the rate of about 30 per cent per year per men employed. They also show that over a period of five years practically 100 per cent of the men employed, or their families, are served by these hospitals, so that it is reasonable to assume that for an investment of approximately $30 to $36 per year a miner and his family are insured against a probable outlay ranging from $100 to $200 per year.

The service rendered by these hospitals is a boon to any mining community and a godsend to many individuals within it. It should therefore be one of the major factors of any medical program that coal operators provide some form of hospitalization, either in company-owned hospitals or in outside hospitals wherein these arrangements can be organized in such a manner as to be profitable to the employee and his family, to the hospital, the medical profession and the community as a whole, and therefore of value to the employer.

We recommend that every operator address himself to this particular problem, which in recent years has become so extremely important that the United States Public Health Bureau has in a recent survey found that 52 per cent of the population did not receive any service of any kind, surgical, medical or otherwise; 79 per cent received no attention from dentists; 89 per cent enjoyed no health examination or prevention medical service of any type and 62 per cent of the country's entire population received no medical, dental or eye care of any description.

Consolidation Coal Company Program

In the Consolidation Coal Co. we have addressed ourselves to the problem of keeping the miner and his family well because we have found by experience that it costs less to keep them well than it does to treat them when they become sick.

When our medical program was launched, 90 per cent of our medical staff was made up of contract doctors; i.e., independent doctors who served our various mining communities for the check-off paid to them over the payroll by our employees. There was such general dissatisfaction with this plan that, at the request of the employees in several divisions, we reorganized our program. We immediately developed a plan under which young doctors who had been graduated from recognized medical schools and who had served interneships of at least two years were assigned to the task of keeping the miner and his family well. Benefits of the new plan began to show immediately.

Following the inauguration of the program, a standard list of 133 drugs was adopted for all of our medical units. By centralized and concentrated purchasing in wholesale lots six times per year, we were able to save approximately $10,000 in the first year under the new plan. The physical layout, physical and technical equipment of the offices, and the general procedure of our medical units also have been standardized, so that today every one of our medical units rates far above the average in appearance, equipment, etc.

A program of physical examination of applicants for employment inaugurated at the same time has assisted the operating units in weeding out employees who, because of disease or other infirmities, might automatically become liabilities in the matter of accident, compensation and group insurance. In the early years of this examination program it was not uncommon to reject 30 to 35 per cent of the applicants for employment. Today, normal rejection averages approximately 22 per cent, and the physical status of our employees has improved directly with the per-man-day improvement in production.

An authentic duplicate record is maintained in each medical unit, showing home and office calls, with name of patient, age, race, color, married or single, nature of disability and any other pertinent information that will assist the doctor, nurse or hospital in the follow-up of the case.

The public health nurse maintains a record for every family, indicating the general physical condition of each member of the family, dates of immunization for all contagious diseases, physical defects in limbs, eyes, ears, nose, throat, teeth, weight, etc. This record when used in connection with periodical physical examinations is extremely valuable to the company physician when he is called to treat any member of the family. The nurse's record of prenatal and post-natal care of mothers is also valuable to the doctors and our prenatal and post-natal clinics are directly responsible for an extremely low mortality and morbidity among mothers and babies.

In addition to our medical units, serving approximately 12,000 people, we have a modern 60-bed hospital fully equipped to perform any type of major or minor operation. In this hospital we are equipped with X-ray and pathological laboratories, which have saved the company considerable money and have been of invaluable service to our employees and their families. Our pathological laboratory has materially added to our service in the hospital as well as in the medical units, in that it has facilitated the diagnosis of contagious diseases. It is now possible to isolate through smears and culture bacteria that cause many of the acute infectious diseases, which is of great importance to the physician in determining a specific diagnosis, and in turn administering specific treatment, particularly in the form of serums.

We believe that one reason why we have been particularly fortunate compared to other Kentucky communities in the incidence and the results of treatment to cases of meningitis is largely because of our ability to secure through efficient laboratory findings the specific germ responsible for the specific type of meningitis and to prescribe a specific treatment, which has resulted in cures in all cases. Our laboratory also enables us to isolate such bacteria as tuberculosis, syphilis, typhoid, gonorrhea, pneumonia and Streptococci infections.

A further advantage afforded by the laboratory is in analyzing an acute or inflamed eye, which the patient may claim has been caused by an occupational injury. Upon examination of eye smears, we find that in many cases the patient is suffering from an acute gonorrheal infection, which might have resulted in an unreasonable cost to the company, probably the loss of the eye to the patient, and possibly other infected cases in the community.

As a therapeutic agent our laboratory serves in two capacities; namely: (1) in the preparation of autogenous vaccine, the matching and typing and preparation of blood for transfusions, and (2) in the accurate compounding or mixing of drugs and chemicals. For example: (1) for a patient afflicted with boils, a culture can be made from the specific causative bacteria and a vaccine made that is often beneficial; (2) our glucose and saline solution are accurately weighed and prepared in our own laboratory. We have used these preparations as furnished by our laboratory for the past eight years and results have been unusually successful.

The second feature of our program involves physical examinations. Over a period of five years we have found this procedure extremely valuable in the supervision of compensation and group insurance, and generally beneficial to our communities as a whole, in that it eliminates the possibilities of employing men afflicted with tuberculosis, venereal disease, etc., and likewise eliminates from the possibility of accident or subsequent social problems those applicants who have defects such as defective vision, hernia, potential hernia, hypertension, heart disease, trachoma or trench mouth.

About seven years ago our company inaugurated a group insurance plan wherein our employees, for a nominal monthly premium collected over the payroll, were protected in the following types of coverage: Life insurance with a minimum of $1000 and a maximum of $2000 for all employees; accident and health insurance wherein an employee would receive for an occupational or nonoccupational accident or illness $12 per week for a period of 13 weeks. In the administration of this program our company physicians and personnel managers are directly responsible for the supervision and payment of death, accident and health benefits.

LIMITATION OF ILLNESS

In our administration of this program we have been able over a period of seven years to determine within reasonable limits the average days of disability covering all types of occupational or nonoccupational accidents and illness.

We have two operating divisions, in which there are approximately 3000 workers. Our medical set-up and personnel, our medical and public health program, in both operating divisions is practically identical. The working conditions within the mines and house-plant conditions outside the mines are practically identical. Our personnel as to native Americans, colored and foreigners, is balanced approximately the same in both operating divisions, with native Americans predominating at about 80 per cent of total personnel.

Notwithstanding that all of these conditions and circumstances were approximately parallel and therefore comparable, we have found during the past several years that the incidence of illness in one division is practically twice that of the other division. We arranged to have an outside public health expert visit all of our operations and interview our doctors individually and collectively, and particularly to investigate conditions in the two divisions to see wherein our accident and health experience was badly out of ratio. It was found that our medical and hospital program and facilities as well as the personnel rated very highly and compared most favorably with the other division wherein our experience had been most favorable. It was also determined by laboratory tests that our milk and water supply could not be questioned in any manner. Our hospital records, however, indicated that 52 per cent of our operating cases showed the presence of round worms or pin worms, particularly where such operations involved abdominal or intestinal operative procedure. The public health expert concluded that some fundamental or inherent physical or geographical cause was responsible for the apparent lowered resistance of the employees in one division as compared to these factors in the employees in the other division that was used as a measuring stick or a basis of comparison in our survey. Accordingly, a survey was initiated wherein the stools or feces of a cross section of our employees were taken and analyzed in our company hospital laboratory.

In this particular community, comprising approximately 3000 workers and a population of approximately 12,000, we have found to date in the laboratory tests of 1000 individuals that the following condition prevails: The presence of parasites is indicated in 70 per cent of the tests. So far these parasites are of the following type: intestinalis, round worm, pin worm and hookworm, with the hookworm predominating; in fact, the hookworm is found in 40 per cent of the persons afflicted.

While we have not at this date quite completed our survey, we have gone so far as to definitely conclude that the reason for the apparent lowered resistance among our employees in this particular division and the primary reason why the incidence of illness, particularly having to do with the respiratory organs, is so abnormally high can be attributed largely to the presence of parasites, as heretofore indicated. We now know that we have a general public health problem to solve in this particular community, and that the problem can be met not alone by proper treatment and medication but by general education on the part of our medical staff, our public health nurses, and our school system.

We are planning to attack the problem along these lines, knowing that the experience, particularly in the southern states, has definitely shown that the hookworm has been such a menace to the general welfare and public health of communities that well-known foundations, medical associations and State Public Health authorities have addressed themselves to this problem for many years and that as a result of corrective measures the general health and physical stamina of certain types and classes of people in the southern states have been materially improved, with a resulting lowered sickness and death rate.

We cite this experience as a good example of a situation that calls for a progressive medical program materially supported by a competent medical and public health staff. We also cite it as an outstanding situation wherein the promotion of a group insurance program coordinated with a progressive medical program makes it possible, practical and mutually advantageous to employees and their families, and to the company in the working out of a solution or corrective measure designed to improve the general health of workers in the affected communities.

Conserving Human Equipment

We firmly believe that the conservation of health among miners in the coal industry is as important to the industry as is the purchase and efficient operation of its mechanical equipment, which is primarily designed and installed to reduce operating costs.

The selection and proper maintenance of human equipment is equally as important as the selection and maintenance of mechanical equipment. But don't make the mistake of leaving it all to the doctor. He needs your interest, support and supervision of certain phases of mining and community life that are necessary to his success and to the success of any public health program you may organize. And do not forget that he and your public health nurse, if you have one, can help you to get the company across to your employees, which in these days of complex labor relationships is a man's size job in itself.

Prospecting for Anthracite by the Earth-resistivity Method

By Maurice Ewing,* A. P. Crary,† Joe Webb Peoples,‡ Junior Members A.I.M.E. and James A. Peoples, Jr.§

(New York Meeting, February, 1936)

The purpose of this paper is to present the results of the application of the earth-resistivity method of subsurface investigation to the problem of locating seams of anthracite coal beneath a mantle of soil and subsoil. Details are given of the successful use of this method in locating the outcrops of anthracite seams in actual prospecting. The presence of the outcrop of an anthracite seam is indicated at points where electrical resistivity tests showed relatively low readings. The reason for this is that in some localities the resistivity of anthracite was found to be about 0.0001 megohm-cm. compared to 1 to 200 megohm-cm. for shales and sandstones. For two reasons it is believed that the data summarized in this paper should be of particular interest. In the first place, most of the interpretations of the resistivity survey have been checked by subsequent drilling and some by the removal of the overburden and the exposing of the anthracite seams. It is infrequent that the opportunity for checking results so thoroughly is given. In the second place, the fact that recent commentators[1,2,3] on the application of geophysical methods to mining have concluded that direct prospecting for anthracite and other types of coal is not promising makes the good results obtained more significant.

So far as the authors can learn, the only previous experiment with electrical methods in the Pennsylvania anthracite field was one made by Sherwin F. Kelly in 1921, when he used the self-potential method at Wilkes-Barre[4]. The profile he made does not show a very marked correlation with the geologic cross section. Kelly points out that stray currents from electric motors in the mines undoubtedly affected his results, but the experience of the present authors with stray currents from mining machinery suggests the possibility that the potentials measured by Kelly were due almost entirely to this effect.

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[1] References are at the end of the paper.

The surveys described herein were made between Sept. 1, 1934 and Aug. 1, 1935. All of the surveys were in the western middle anthracite

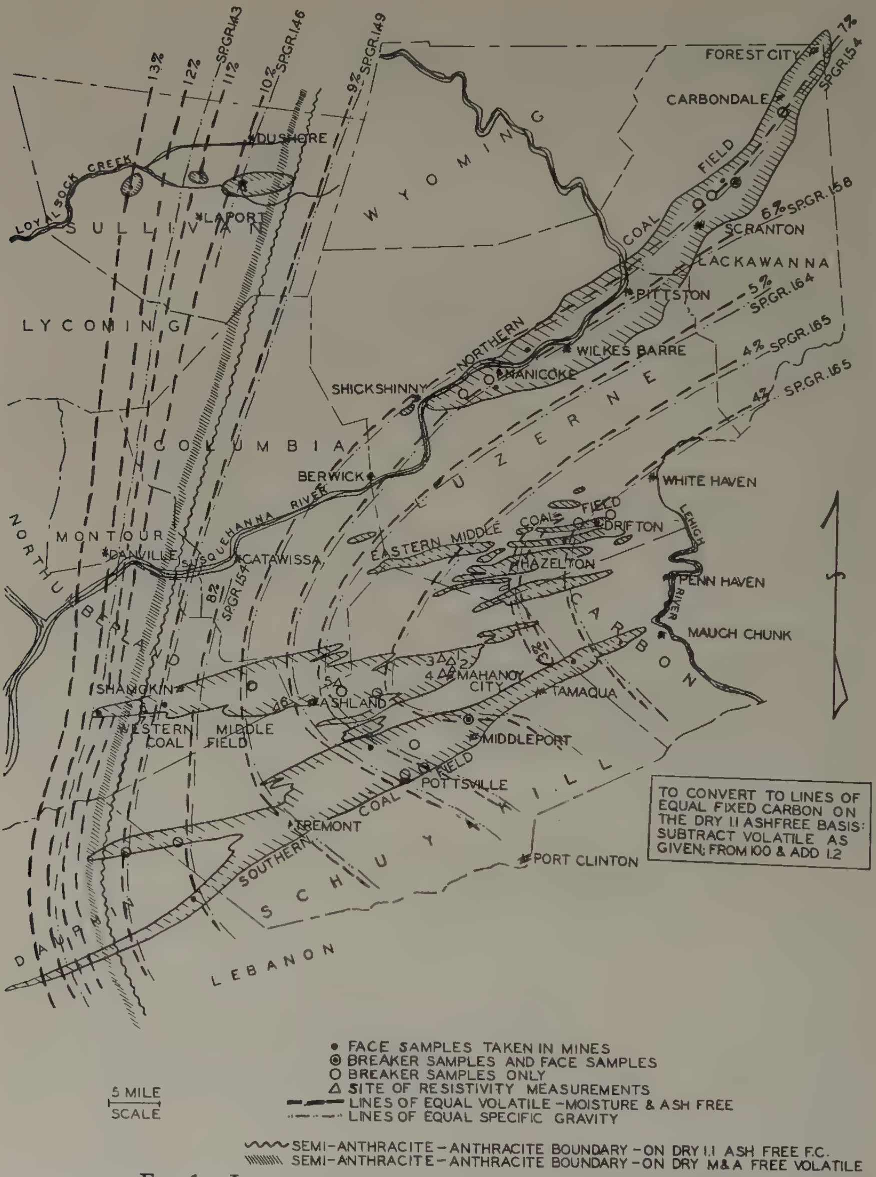

FIG. 1.—LOCATIONS OF FIELD RESISTIVITY MEASUREMENTS.
This map is copied from H. G. Turner: *Trans:* A.I.M.E. (1934) **108**, 330–344.

field and most of the work was in the Mahanoy City district. The locations of the different field projects are shown on the map in Fig. 1.

General Geology

Structurally the western middle anthracite basin is a complex synclinorium, consisting of several smaller basins or synclines separated by anticlines or thrust faults. The structure as recognized toward the end of the last century has been described and illustrated in the reports of the Second Pennsylvania Geological Survey[5,6,7]. The width of the synclinorium measured from the outcrops of the Buck Mountain seam on opposite sides of it is a little over 9000 ft. in the vicinity of Mahanoy City. In the western end of the synclinorium, faulting is rare, but in the eastern part, in the vicinity of Shenandoah and Mahanoy City where the resistivity surveys were made, the presence of thrust faults has greatly increased the complexity of structure. The folding was intense, and vertical and overturned beds are common. In 1886 Frank A. Hill wrote: "There is probably no area in the Anthracite Region which contains so much that is interesting in structural geology as that embraced by Mine Sheet No. II." (Mine Sheet No. II covers the area from Mahanoy City to the west side of Shenandoah Ref. 5, p. 987.) Later mining has shown the structure to be even more complex than described by Mr. Hill and the other geologists of that survey. Near Mahanoy City from south to north the following basins or synclines may be distinguished: Mahanoy, Middle Mahanoy (or Mahanoy City), Ellangowan, Yatesville and Knickerbocker, the last two together forming what was designated by the Second Pennsylvania Survey as the Shenandoah Basin[6]. Local folds and faults not prominent enough or constant enough to have names are common.

The anthracite seams of the western middle field are shown in Table 1. The names are used except in the Shamokin district, where the seams are referred to by number. The stratigraphic interval between the anthracite seams is seldom more than 150 feet.

Table 1.—*Anthracite Seams of the Western Middle Field*

Number	Name	Number	Name	
16	Tracy	9	Top Split	Mammoth
15	Little Diamond	8½	Middle Split	
14	Diamond	8	Bottom Split	
13	Little Orchard	7	Skidmore	
12	Orchard	6	Seven Foot	
11	Primrose	5	Buck Mt.	
	Little Primrose	4	Lykens Valley	
10	Holmes	3	Lykens Valley	
9½	Four Foot			

In the western middle field a mantle of soil and disintegrated rock varying in thickness from 1 or 2 ft. to 40 ft. overlies bedrock. The thickness of this mantle, which commonly is referred to in the region as "wash," probably averages about 15 ft. in the areas studied. The purpose of the resistivity surveys was to locate the intersection of the coal seam with the mantle; or, according to the terminology followed in this paper, the "crop" of the seam. In certain cases where more specific information was desired and was feasible the crop of the bottom slate and top slate of the seam were located.

The Mahanoy region has been intensively mined for more than 70 years. Most of the seams have been worked near the surface, but as this work was for the most part in the early days of mining, complete extraction was not obtained. The coal left near the surface is being recovered in many places by stripping operations. The old workings were not always completely and accurately surveyed and consequently cross sections prepared from the mine-map data were not always accurate representations of actual detail conditions. The methods of prospecting commonly used to gain additional information are: (1) test shafts and drifts, and (2) diamond or churn drilling. Even though the area studied has been mined intensively for many years, virgin bodies of coal, not heretofore known, are yet to be found, as illustrated by the Orchard spoon shown in Fig. 7.

The cross sections exhibited with the resistivity profiles were constructed from mine maps and churn-drill records. These have not been altered by the authors to fit the resistivity data, with the exception of Fig. 16. They have been altered only when churn-drill records or other new information has been secured. In a number of cases where the resistivity profiles did not fit the original cross sections, later drilling has shown the cross section to be in error. In these instances the sections were changed to accord with the drill records, which, as it happened, were in agreement with the resistivity profiles. Lack of agreement between the resistivity profiles and the cross section, therefore, is not significant unless the structure has been proved by drilling. The discrepancies between the resistivity data and positively known structures are very few in number and small in size.

METHOD AND APPLICATION

The method followed is that worked out by Wenner[8] and by Gish and Rooney[9] and followed by many workers since. Four electrodes are driven in the ground in straight line at equal intervals. The spacing between adjacent electrodes is taken as a. An electric current, I amperes, is passed between the two outside electrodes, C_1 and C_2, and the difference in potential, V volts, between the inner electrodes, P_1 and P_2, is measured. (See Fig. 2.) The apparent resistivity ρ, in megohm-centimeters, is given by the equation

$$\rho = 2\pi a \frac{V}{I} \times 10^{-6}$$

where a is the electrode spacing expressed in centimeters. The resistivity of a material in megohm-centimeters is numerically equal to the resistance, measured in millions of ohms, of a specimen of the material 1 cm. long and 1 sq. cm. in cross-sectional area.

There are really only three quantities that may be varied in a measurement of apparent resistivity: (1) the position of the station, which is taken as the midpoint of the line of electrodes; (2) the electrode spacing a, and (3) the azimuth of the line of electrodes.

Following the usage of Hubbert[10], the following field techniques are differentiated:

1. *A traverse profile* is a series of observations taken along a line of traverse while both a and the azimuth are kept constant.
2. *A longitudinal traverse* profile is a traverse profile wherein the line of electrodes is parallel to the line of traverse.
3. *A transverse traverse profile* is a traverse profile taken with the line of electrodes perpendicular to the line of traverse.
4. *A circular profile* is obtained by rotating the line of electrodes about a given station and reading ρ at equal angle intervals.
5. *A depth profile* is obtained by measuring ρ for successively increased values of a while keeping the position and azimuth constant.

In the work covered in the present report it was found that *transverse traverse profiles*, while slower, gave more dependable and accurate results than *longitudinal traverse profiles*. Nearly all the profiles figured herein are either transverse traverse profiles or depth profiles.

APPARATUS

In all resistivity measurements reported here, except project 5, a 300-volt direct-current generator driven by a 32-volt motor was used as the source of current. The current, usually between 0.1 and 0.2 amp., was introduced into the earth by means of iron stakes driven about 1 ft. deep, one stake at C_1 and one at C_2. A commutator K_1 (Fig. 2) on the generator shaft served to reverse the direction of flow of current at the rate of about 30 reversals per second. The value of the current supplied by the generator G_1 was read on the ammeter A.

Two different arrangements were used for measuring the voltage between the potential electrodes P_1 and P_2 (Figs. 2 and 3). The first (Fig. 2) is a special voltmeter arrangement, in which K_2 is a commutator on the same shaft with K_1, which serves as a rectifier for the alternating potential that exists between P_1 and P_2. G_2 is a galvanometer, which indicates zero current when the sliders on the rheostats R_1 and R_2 are properly adjusted, and V_2 is a voltmeter that gives the potential between P_1 and P_2. With this arrangement, it was necessary only to adjust the rheostats until the galvanometer indicated zero current, and to read voltage and current from V_2 and A respectively. The second arrangement for measuring potential utilized a voltage divider D, connected, as shown in Fig. 3, directly across the current leads. The voltage divider had a total resistance of 10,000 ohms and divided the total applied voltage into 1000 equal parts. For taking a reading, the potential electrode P_1 was connected into the circuit by means of the switch S and the voltage divider D was adjusted until the galvanometer G_2 indicated zero. The reading of the voltage divider then gave the *fraction* of the total applied voltage V_1 existing between C_1 and P_1. Repetition in this procedure

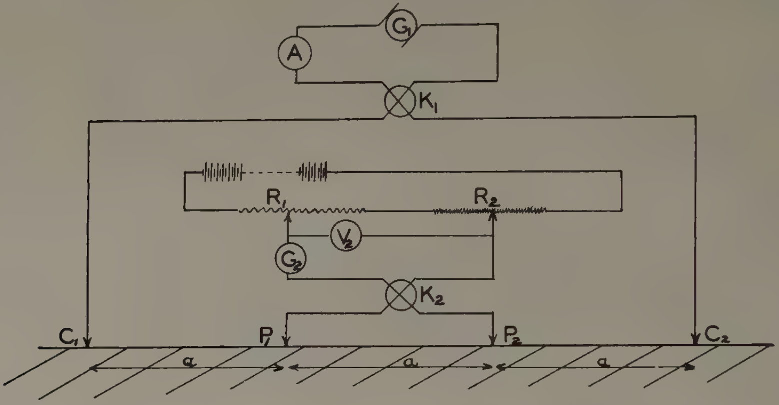

FIG. 2.—DIAGRAM OF CIRCUIT USED IN RESISTIVITY MEASUREMENTS.

G_1 Generator
A Ammeter
K_1 Current commutator
R_1 and R_2 Rheostats
V_2 Voltmeter
G_2 Galvanometer
K_2 Potential commutator

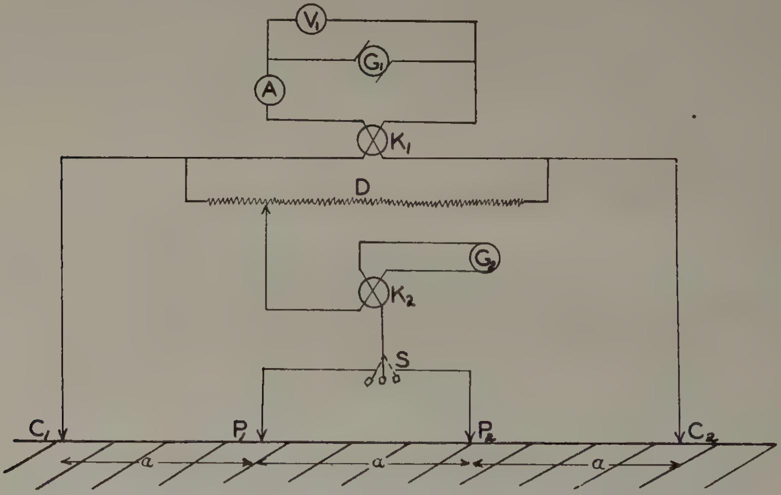

FIG. 3.—DIAGRAM OF CIRCUIT USED IN RESISTIVITY MEASUREMENTS.

G_1 Generator
A Ammeter
K_1 Current commutator
D Potential divider
V_1 Voltmeter
G_2 Galvanometer
K_2 Potential commutator
S Switch

with P_2 instead of P_1 in the circuit gave sufficient data for computing the potential difference between P_1 and P_2, and hence the "apparent resistivity." The system shown in Fig. 3 is inferior to that of Fig. 2, and was used only at times when some of the apparatus in Fig. 2 had to be surrendered for other uses.

In both systems the commutators K_1 and K_2 eliminated all effects due to stray currents in the earth. This was of utmost importance because the differences in potential due to these stray currents were frequently found to be several times as great as those due to the current passed from C_1 to C_2. The commutators had to be built after the work was started, as the possibility of stray currents had not been taken into account in the original plans.

In calculating the resistivity it was necessary to make an instrumental correction introduced by the commutator K_1. This commutator opened the circuit for a short time between reversals, and while the circuit was open the current was zero, of course. Thus the ammeter A indicated a current somewhat smaller than the true current that flowed while the circuit was closed, a sort of average between this current and zero. To determine the correction for this effect the resistivity apparatus was used to measure the resistance of a standardized resistance instead of that of the earth. The measurement of the standardized resistance was repeated from time to time during the course of the work and served as a very convenient check upon the proper functioning of the entire apparatus.

On the survey of project 5 the first arrangement for measuring the potential was used with the modification that the heavy motor-generator set was replaced by four radio "B" batteries as a power supply. This was done for the sake of portability, since the terrain was very rough and all the equipment had to be carried by hand. This change took away the commutators, so it became necessary to read the stray current at each station and to make a correction. To work without a commutator set would not have been feasible if the stray currents had not been steady and small.

FIELD TECHNIQUE

Since traverses were at first made across the crops of seams that had been located accurately by drilling, it was comparatively simple to determine the most suitable field technique at the outset. Experimentation with both longitudinal and transverse traverse profiles, and with various electrode spacings, showed that transverse traverse profiles, though slower to carry out, were so much more reliable and accurate than longitudinal traverse profiles that the latter were not used thereafter. An electrode spacing somewhat greater than the depth of mantle material was found to be best, which is in accord with the general assumption that the effective depth of penetration of the currents is approximately equal to the electrode spacing. Since the average depth of the mantle was 12 to 15 ft., a 20-ft. electrode spacing was used, except where depth profiles or drill holes indicated thicker mantle, when 30-ft. spacing was used.

Circular profiles were needed only occasionally, since the strike ordinarily was known. Depth profiles were made over many low-resistivity points for gaining additional information. For all depth profiles the line of electrodes was parallel to the strike of the strata. Profiles were usually made at intervals of 200 ft. Many times, however, the line of traverse had to be offset to avoid large mine breaches. Stations were usually 10 ft. apart. The measurements were made with a metallic tape but stakes located by a transit survey every 100 ft. prevented any cumulative error.

A field party consisted of four men, but many surveys were made with only two and three men.

Field Studies

Project No. 1

This project represents a survey to locate the crops of several seams of anthracite in part of the Yatesville Basin east of the Mahanoy City-Brandonville highway, and about one mile north of Mahanoy City, Pa. Five transverse traverse profiles were made at intervals of 200 ft. The results of this survey are shown by a resistivity map in Fig. 4 and five profiles, Figs. 5 and 6.

When this survey was made the structure of this part of the Yatesville Basin was imperfectly known because very little mining had been done. Therefore the original cross sections made from mine maps were poor. However, the Holmes syncline and the south-dipping Top Split of the Knickerbocker Basin on the 92 + 00 E. line had been prospected by churn-drilling. This was chosen as a place to test the method against a known structure. The short profile along 92 + 00 E. across the Top Split and Holmes seams shown in Fig. 5 is the result. Minima were found over the coal in both tests, therefore this was considered a satisfactory test of the method. It is to be noted that the minimum above the Top Split crop is about 0.12 megohm-cm. while that above the Holmes basin is 0.02 megohm-cm. This difference was explained at the time of the survey as due to the high resistivity of the sandstone found below the Top Split seam. When the Top Split seam was stripped, however, it was found to be somewhat thinner and steeper than shown on this cross section.

There is no electrical indication of the north-dipping Holmes seam along the 92 + 00 E. line, which probably means that the seam is pinched and is very thin on this section line. About 1000 ft. farther west, where it has been exposed, it is only 2 or 3 ft. thick. To the east it is known to be a good seam; i.e., 10 to 20 ft. thick. On the original cross sections the Holmes seam was shown cropping where Top Split is shown in Fig. 5. The Top Split seam has since been stripped for several hundred feet along its strike and found to agree with the electrical findings rather than with the original sections.

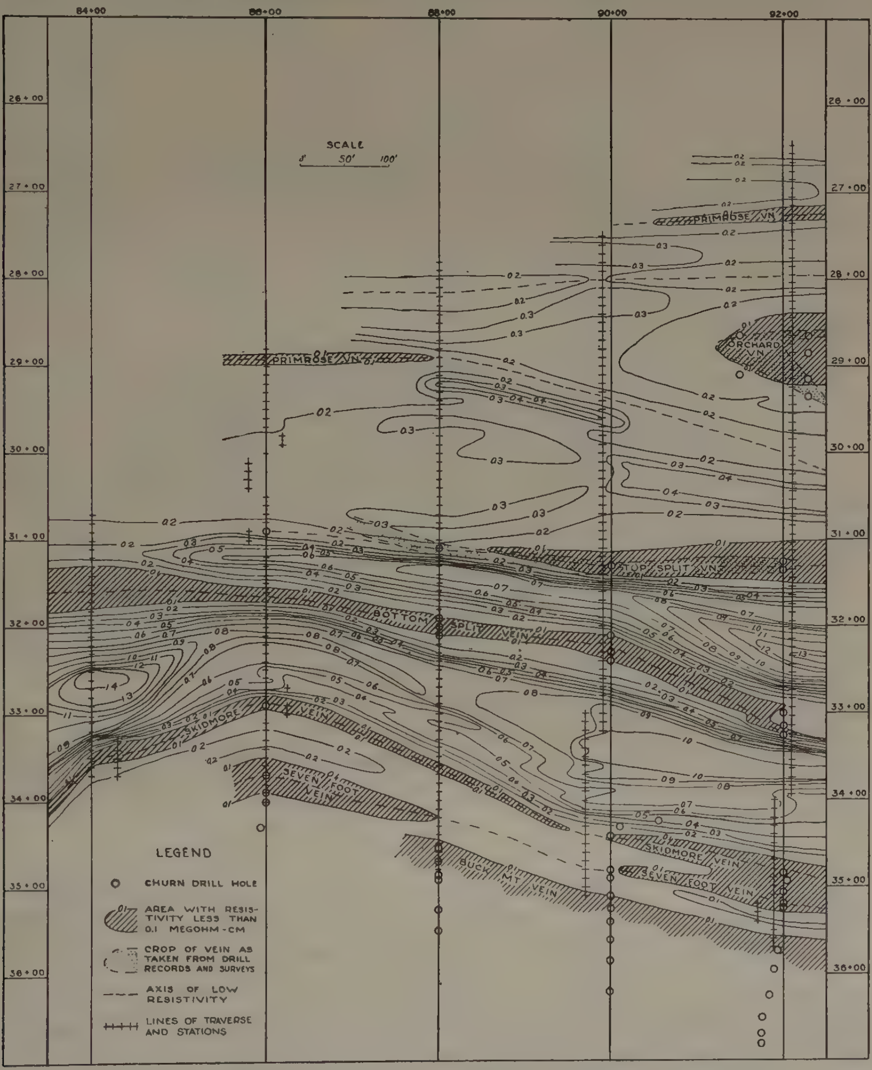

FIG. 4.—RESISTIVITY MAP OF PROJECT 1. NORTH AT TOP OF PAGE. Electrode spacing 20 ft. E.-W. Resistivity in megohm-centimeters.

The Top Split seam was located by the resistivity survey and the drill holes were planned on the basis of this survey. On the original cross section, the crop of the Top Split seam was over 100 ft. farther south. The next two seams located were the Middle Split and Bottom

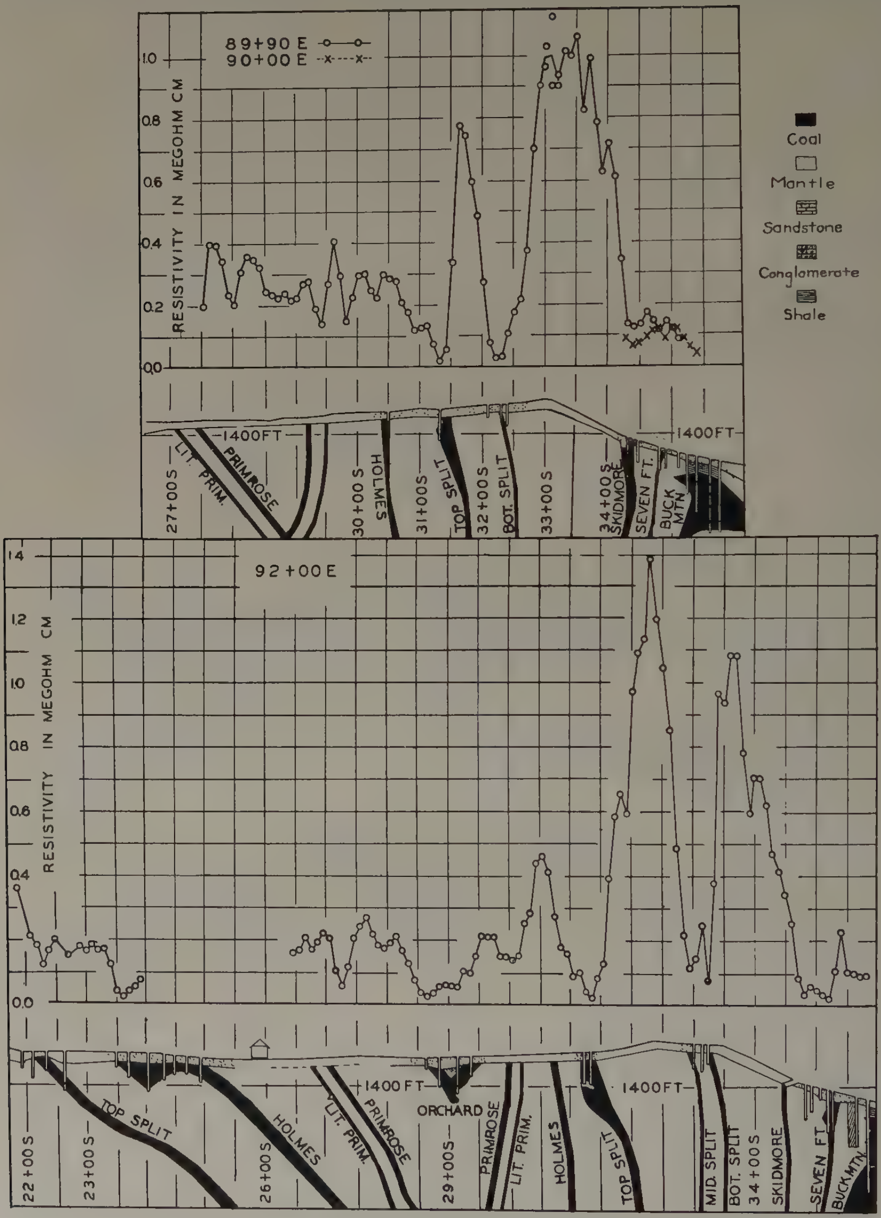

Fig. 5.—Sections for project 1. North at left.

Split. The presence of the former in this part of the Yatesville Basin had not been previously known. It was not found farther west. It was not known positively until after the ground had been drilled whether the next minimum to the south represented one or two seams. The Seven Foot was drilled, and a bootleg slope projected 40 ft. west shows the approximate position of the Skidmore. The crop of this seam may be 10 to 15 ft. farther south than shown, since the strike is variable near this section line.

The traverse was not continued to the south because the Buck Mountain anticline was thought at the time of the survey to be too deep to be of interest. The last four readings were about 0.1 megohm-cm. and their low value is probably due to the shale more than to the coal. The crop of the Buck Mountain anticline was located by the resistivity survey on the 88 + 00 E. line and after it had been drilled it was drilled also on the 92 + 00 E. line, as shown in Fig. 5.

The profile along 89 + 90 E. is very similar to that along the 92 + 00 E., as would be expected. The Primrose syncline is shown as it was on the original sections, although it is obvious from an inspection of the profile that the syncline should be farther south. The original cross section had little basis, since the Primrose had not been mined at this place, but the sections have not been changed unless drill holes or other new data were available. No Orchard seam is present, though the center of the Yatesville syncline is indicated by a drop to 0.2 megohm-cm.

As on the previous section line, there is no clear indication of the Holmes seam. The position of this seam on the cross section is based upon an old shafting long since covered up. The small minimum at 30 + 50 S. might indicate the crop of the seam, but, if so, the inference is that the seam is very thin or under considerable cover of mantle.

The minimum at the crop of the Top Split is sharp, as is the one at the crop of the Bottom Split. Between the two is a high-resistivity area as on the 92 + 00 E. section line. The values were not quite as high here as on the previous line, which may be due more to topography and to drainage conditions than to differences in the rock beneath the mantle. On this line 89 + 90 E., higher resistivity readings were obtained between Bottom Split and Skidmore seams. The resistivity between the Skidmore and Seven Foot seams is remarkably low. The drill holes on the Top Split, Bottom Split, and Seven Foot seams were located from the resistivity profile.

The readings indicated by the crosses on Fig. 5 were taken eight months after the main traverse was made and after the Buck Mountain anticline had been drilled on the 88 + 00 E. line (Fig. 6). The last two readings may have been influenced by culm left on the ground when a culm bank was removed.

The profile along 88 + 00 E., like those just described, shows fairly good indications of the Primrose, little indication of the Holmes, marked minima above the crops of the Top Split and Bottom Split seams, and small minima above the Skidmore and Seven Foot seams. The two seams that were drilled were found where predicted from the resistivity data. This traverse was continued south of the crop of the Seven Foot seam and another marked minimum was obtained as low as 0.02

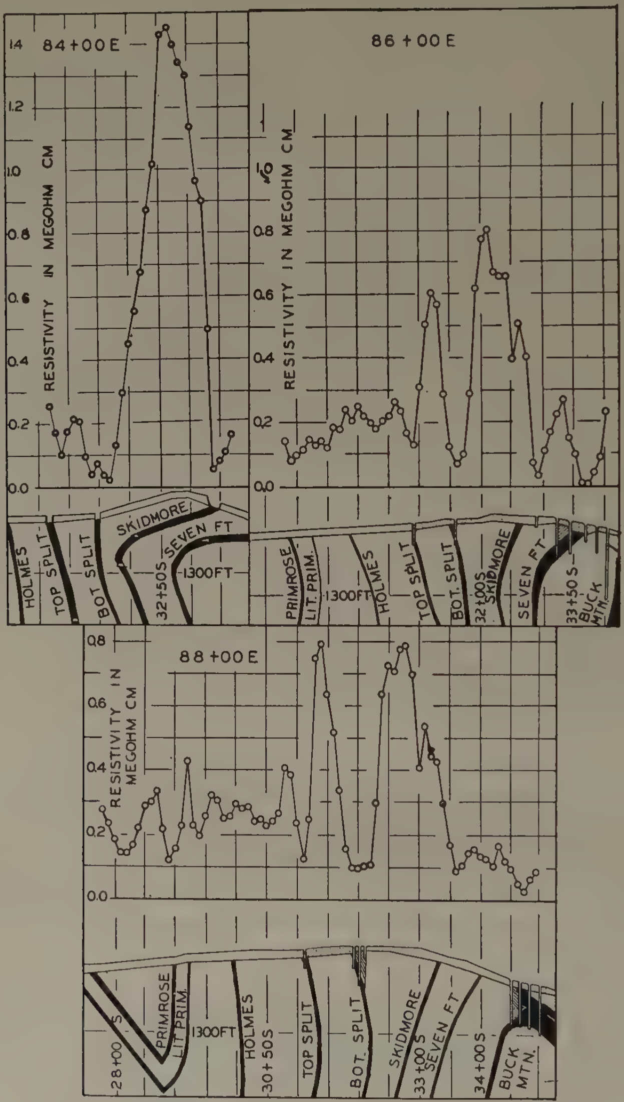

FIG. 6.—SECTIONS FOR PROJECT 1. NORTH AT LEFT.

megohm-cm. This was drilled and the peculiar swell on the crest of the Buck Mountain anticline was discovered. Following this, the Buck Mountain anticline was drilled on sections 90 + 00 E., 92 + 00 E. and 86 + 00 E., as shown in Figs. 5 and 6.

On the 86 + 00 E., marked minima were obtained above the crops of the Top Split, Bottom Split, Skidmore and Seven Foot seams. The exact position of the crop of the Skidmore is not known; the one hole drilled failed to locate it. It cannot be far north of this hole, however, for the seam is to be seen in a breach a short distance west from this section line.

The lowest resistivity above the crop of the Seven Foot was only 0.01 megohm-cm., but a drill hole at this point penetrated 10 ft. of black dirt and no coal. The lowest readings were obtained where the black dirt extends almost to the surface, for it is as much better a conductor than ordinary mantle rock as coal is than the country rock. In predicting the crop of a dipping seam, the black dirt must be considered, and it is best to drill down the dip from the lowest reading obtained. Much higher resistivity was noted between the Skidmore and Seven Foot seams on this line than on previous section lines. This is thought to be due largely to the topography and drainage conditions.

The profile along 84 + 00 E. shows the greatest contrasts in resistivity found on any traverse in the anthracite region. The rock giving the high reading is sandstone but the topography and good drainage conditions probably affected the result considerably. The highest resistivity was found near the nose of the hill. The hill is well drained and the bedrock is probably rather dry.

The low resistivity between the Top Split and Bottom Split as contrasted with that obtained between these seams on the other section lines described is noteworthy. How much this is due to changes in thickness and lithology and how much to drainage and topography is not known.

The resistivity map in Fig. 4 shows the results of this survey in another way. The low-resistivity areas above the crops of the Bottom Split and Top Split seams are very marked because of the high resistivity between them. The Orchard seam also shows up very well. The actual crop of the bottom and top of the Top Split and Orchard seams as determined by drilling and surveys after the overburden was removed is shown. Since the traverses could not extend very far to the north on the 86 + 00 E. and 88 + 00 E., and 89 + 90 E. lines, there is a little question as to how the resistivity contours should be drawn near the north edge of the map. This affects only the Primrose seam. The low-resistivity areas defining the crops of the Seven Foot and Skidmore seams are fairly distinct. The traverses were not extended far enough south for the 0.1 megohm line to be closed.

No good indication of the Holmes seam was obtained on any of these profiles, probably because the seam is very thin or faulted out in this area. At about 300 ft. east of the 92 + 00 E. section line, the Holmes seam, which is exposed in a breach, is cut by a thrust fault. In the neighborhood of 78 + 00 E. where the Holmes has been exposed it is only 2 ft. or so thick. The fact, therefore, that no indication of the crop was obtained by the resistivity survey is not surprising.

The results of this project may be summed up as follows: In an area where the structure was imperfectly known, the crops of all seams thought to exist (with the exception of the Holmes) were located and where the low-resistivity points have been drilled the seam has been found with few barren holes even though many of the seams were thin and often vertical. The Orchard seam (Fig. 7), which was not known to be present, was located. The Buck Mountain anticline, the top of which was not thought to be nearer than 40 ft. from the surface, was found just below the mantle on one section line. This survey, therefore, was very successful in locating the crops of seams in this area as shown by subsequent drilling and by stripping operations.

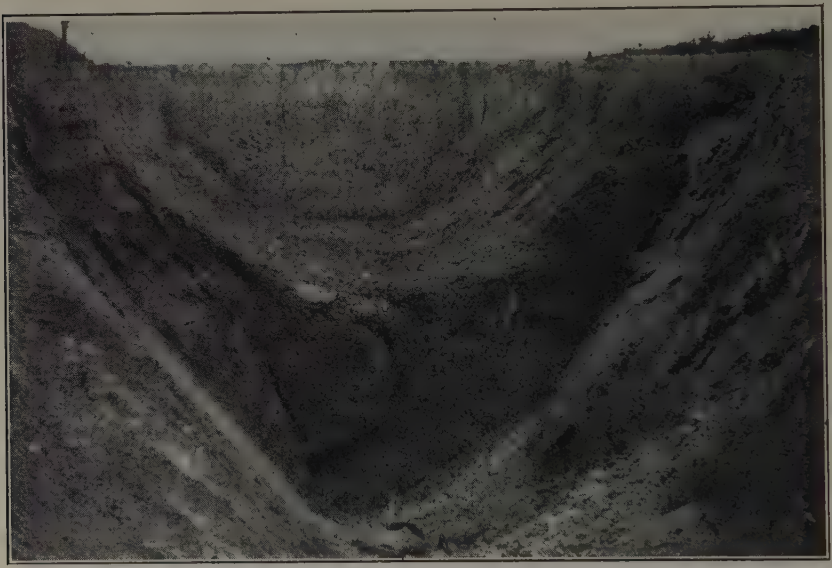

FIG. 7.—SYNCLINE IN ORCHARD SEAM NEAR 92 + 00 E. LINE.
Depth to coal in center of syncline is approximately 25 feet. Photograph by E. B. Douglas.

Project No. 2

The purpose of this survey, project No. 2, was to locate the crops of the Bottom Split and Skidmore seams on both limbs of an anticline, just east of the Mahanoy City-Brandonville highway and north of Mahanoy City. The location is shown on the map of Fig. 8. The results are summarized by this resistivity map, five transverse traverse profiles, and three depth profiles, Figs. 9, 10 and 11. When this survey was begun two of these section lines had been drilled and profiles were made along these to test the method on known structures. Some experimentation was done to determine the optimum electrode spacing and the most desirable orientation of the line of electrodes. On Fig. 9 three profiles are shown along 64 + 00 E., a transverse traverse profile with 20-ft. spacing, two longitudinal traverse profiles with 20 and 10-ft.

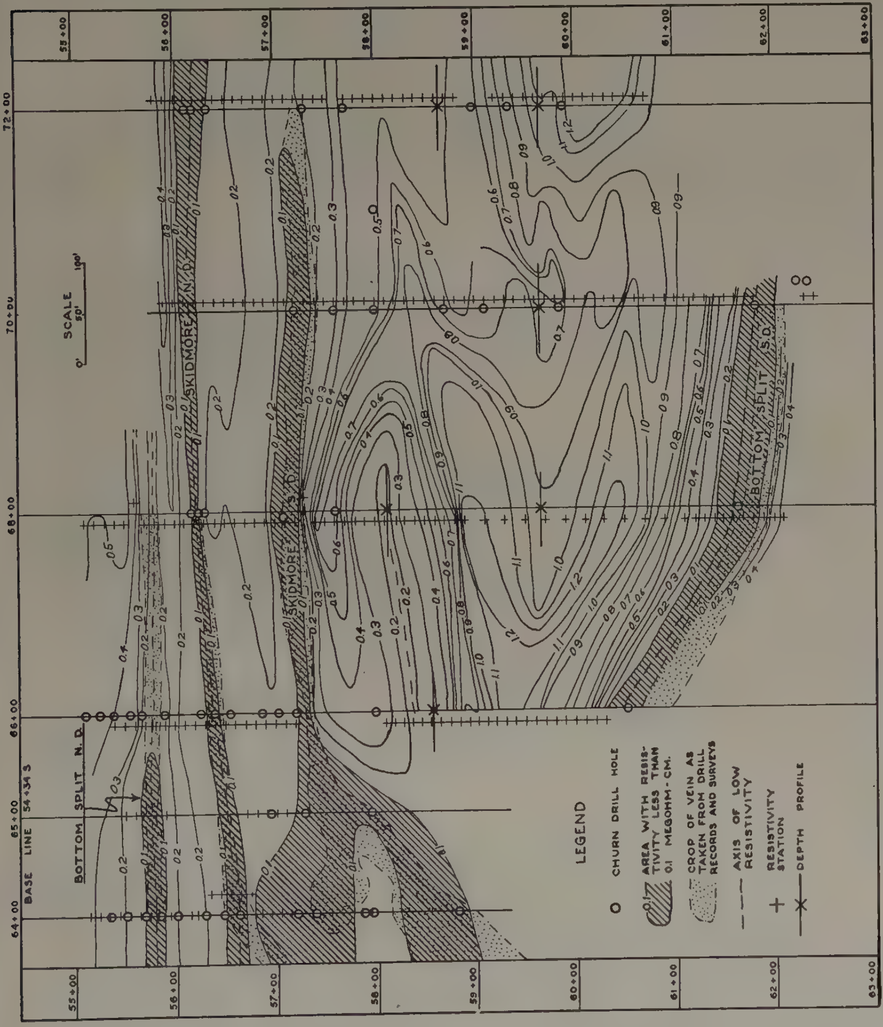

FIG. 8.—RESISTIVITY MAP OF PROJECT 2. NORTH AT TOP OF MAP. Electrode spacing 20 ft. E. W. Resistivity in megohm-centimeters.

spacings respectively. The transverse traverse profile has the sharpest minima and is therefore considered most suitable. The minimum above the Bottom Split seam in Fig. 9 is quite sharp. That above the Skidmore, north dip, is not as distinct. Stripping has revealed that the seam was very thin on this section line, which explains the absence of a sharp minimum. Only these two seams had been drilled at the time the survey was made and the wide low resistivity to the south was a bit of a puzzle until a reexamination of the mine maps showed that a small syncline of Skidmore should have been plotted. This was subsequently drilled as shown. The original sections showed the south-dipping Skidmore, cropping beneath a rock bank, but drilling confirmed the results of the resistivity survey as shown on the cross section in Fig. 9. However, even now, after all the seams shown have been stripped, the width of this low-resistivity area is not completely explained. The Skidmore, north dip, swings sharply to the southwest, which may have affected the results to a slight extent. The mantle is thick, which would tend to give low resistivity. Furthermore, there are indications of the possibility of a very closely folded syncline north of the one figured. In the small basin that was stripped remarkably close folding was noted. This traverse was not only helpful in deciding on the best method to use but furnished some new information, which later was checked in large part by drilling and subsequent stripping.

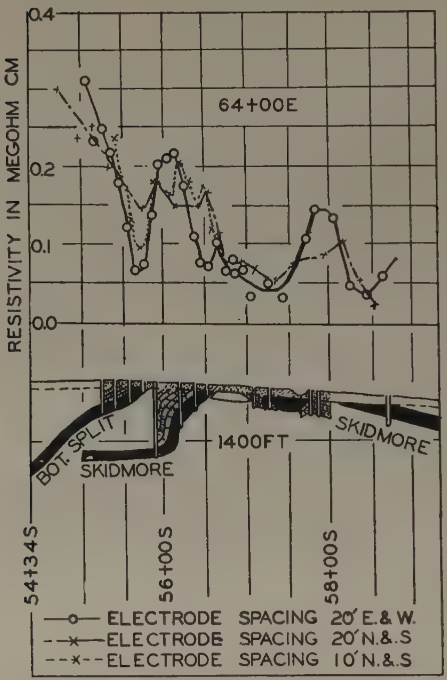

Fig. 9.—Sections for project 2. North at left.

The 66 + 00 E. line had also been drilled before the resistivity survey was made. The Bottom Split, north dip, was missed by the drill. It was suspected to be where the resistivity survey later indicated. The resistivity here dropped to only 0.13 megohm-cm., which at the time was thought to be due to the seam being thin at this place. This opinion has been confirmed by subsequent stripping operations. The minimum above the Skidmore seam, north dip, is fairly sharp. Stripping has shown the seam to be even thinner than was thought. Fig. 12 shows the seam a short distance east of this section line.

The minimum at the crop of the Skidmore, south dip, is not very marked, but a lower reading might have been obtained had not a powder magazine necessitated the omission of two stations. The minimum at 58 + 45 S. is not due to the crop of a coal seam, as shown by the depth profile in Fig. 10. The resistivity increases from the surface to a depth of approximately 70 ft. The low resistivity must be due, therefore, to surface material of low resistance, for the crop of a dipping coal seam would be indicated by a depth profile of the type of that shown in Fig. 14.

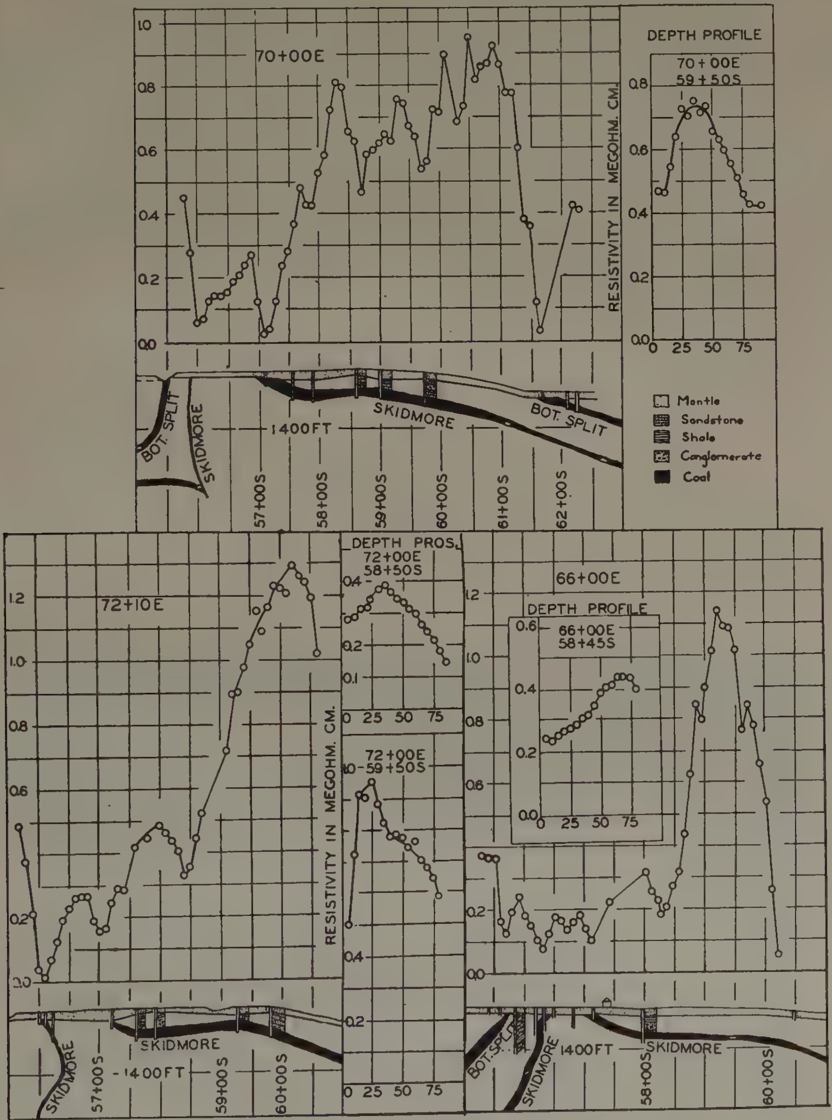

FIG. 10.—SECTIONS FOR PROJECT 2. NORTH AT LEFT.

The last reading obtained was low, possibly owing to the presence of 1 to 2 ft. of culm, and not to coal seam, though the crop of the Bottom Split seam, south dip, must be near.

The resistivity survey of the 68 + 00 E., 70 + 00 E., and 72 + 00 E. lines was made prior to drilling, and the drill holes were located from the profiles. The minimum above the Bottom Split, north dip, on the 68 + 00 E. line is at 0.19 megohm-cm. This reading undoubtedly was affected by the presence of a large mine breach within 20 ft. of each end electrode. A sharper minimum was obtained above the Skidmore seam

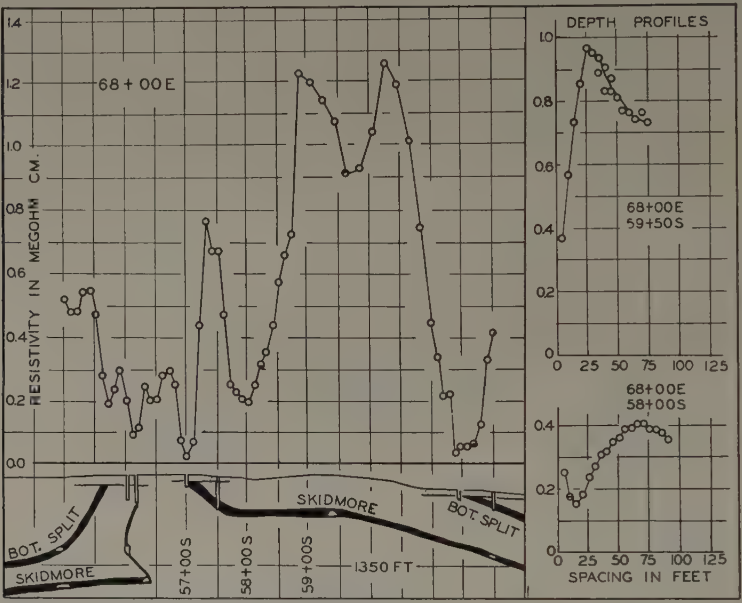

FIG. 11.—SECTIONS FOR PROJECT 2. NORTH AT LEFT.

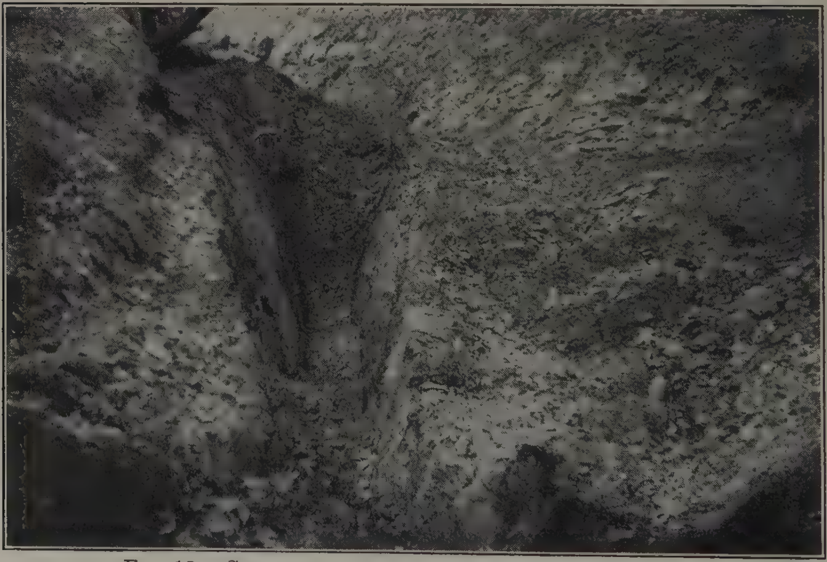

FIG. 12.—SKIDMORE SEAM PINCHED ALONG FAULT PLANE.
Looking west from point near 66 + 00 E. line. Photograph by E. B. Douglas.

although later drilling showed that it was only about 2 ft. thick at that point. The minimum at the crop of the Skidmore, south dip, is very sharp and low. A marked low is to be noted at 58 + 00 S. This minimum has about the same value as that obtained above the Bottom Split, north dip, but the depth profile in Fig. 11 indicates that it is due to surface material of low resistance. Poor drainage conditions, which are apparent, and thick mantle may well account for it.

The minimum above the Bottom Split, south dip, is broad, owing to the low angle of dip of the seam. The drill hole located at approximately the lowest point of the profile struck black dirt, but no coal. This is to be expected where the dips are low. The intersection of the bottom slate and the mantle is often at a few feet downdip from the point of lowest resistivity.

On the 70 + 00 E. line sharp minima were obtained on both dips of the Skidmore and the south dip of the Bottom Split. Two of these have been checked by drilling.

On the 72 + 00 E. the sharp minimum at the crop of Skidmore, north dip, would apparently indicate the crop of a good-sized seam. The drill holes, however, proved that the seam is thin except near the surface, where there seems to be a thickened wedge of coal with 5 to 6 ft. of black dirt above it. This may represent drag along a fault known to have offset the Buck Mountain seam. This is the same fault shown in Fig. 12. The minimum at the crop of the Skidmore, south dip, is not very low and is due to the fact that the seam was very thin and the coal was shelly at this point.

On all of these traverses the high resistivity of the rock above the Skidmore seam is noteworthy. The rock consists largely of medium to thin-bedded, firm sandstone with occasional beds of conglomerate.

Several depth profiles were made along the axis of the Skidmore anticline to determine the depth to the coal. These are shown in Figs. 10 and 11. The resistivity increases with depth to a maximum and then drops. In three cases the maximum was found at an electrode spacing just equal to the depth to coal.

This survey demonstrated that even thin seams could be located by the earth-resistivity method. The depth to coal was predicted with remarkable accuracy in three tests by means of depth profiles. Other depth profiles have not been proved by drilling but they seem more doubtful. The correlation between the low-resistivity areas and the crops of the seams is shown on Fig. 8.

Project No. 3

The purpose of project No. 3 was to locate the crop of the Buck Mountain seam on the north limb of the Knickerbocker syncline (Fig. 1) along several section lines west of the Mahanoy City-Brandonville highway. Transverse traverse profiles were made on six section lines and from these data a resistivity map was constructed and the crop of the bottom slate of the seam predicted. Subsequent drilling has shown the predicted crop to be correct within 10 to 15 ft. The resistivity map is shown in Fig. 13 and the profiles in Fig. 14.

The profile along line 70 + 10 E. shows a minimum at the crop of the seam but the resistivity is over 0.2 megohm-cm., which is much higher than the values obtained at the crops of seams on projects 1 and 2. The higher resistivity apparently is due to thicker mantle and in part perhaps to the high resistivity of the sandstone and conglomerate that overlie and underlie the Buck Mountain seam.

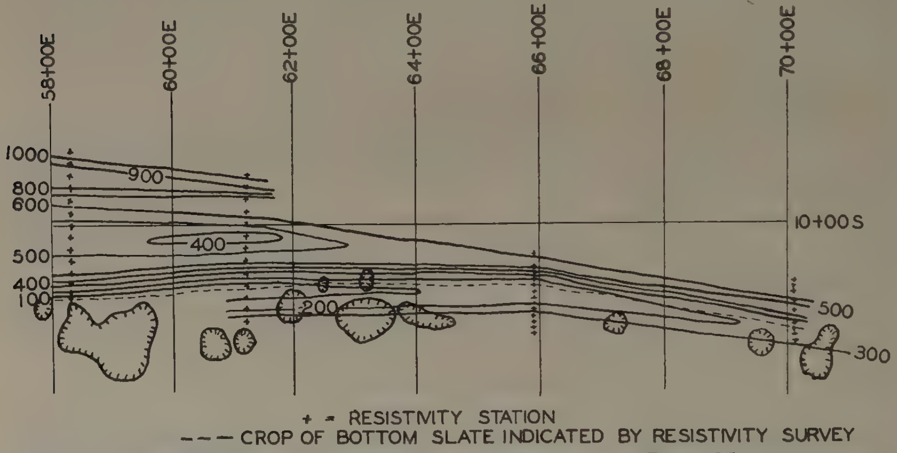

FIG. 13.—RESISTIVITY MAP OF PROJECT 3, SHOWING CROP OF BUCK MOUNTAIN VEIN BETWEEN LINES 58 + 00 E. AND 70 + 10 E. NORTH AT TOP.
Electrode spacing 20 ft. E.-W. Contour interval 100,000 ohm-centimeters.

After a transverse traverse profile with 20-ft. spacing had been made along 65 + 90 E., a traverse with 10-ft. spacing was run along 66 + 00 E., to determine in more detail the limits of the crop. The results with the 10-ft. spacing are due almost entirely to "black dirt" or weathered coal and not to the coal itself, since the coal is 20 ft. deep. The remarkable increase in resistivity to the south is noteworthy and undoubtedly is due to the high-resistance sandstone and conglomerate above the seam.

A depth profile made at 66 + 00 E., 11 + 20 S. (Fig. 14) shows a rapid decrease of resistivity with depth. This is the type of depth profile to be expected above the crop of a dipping anthracite seam. Roman[11] has developed a method for calculating the depth of a discontinuity from depth profiles in two-layer problems. It was thought that the conditions at this place might approximate two-layer conditions closely enough for the application of his method. Using this method the depth to coal was calculated as 16 ft., which an inspection of the drill log shows does not correspond to any discontinuity. This problem is apparently at least a three-layer one, since the surface material, "black dirt" (weathered coal), and coal will each have a different specific resistance. Furthermore, the vein is narrow enough so that effects from the top and bottom slates must be large for the longer electrode spacings.

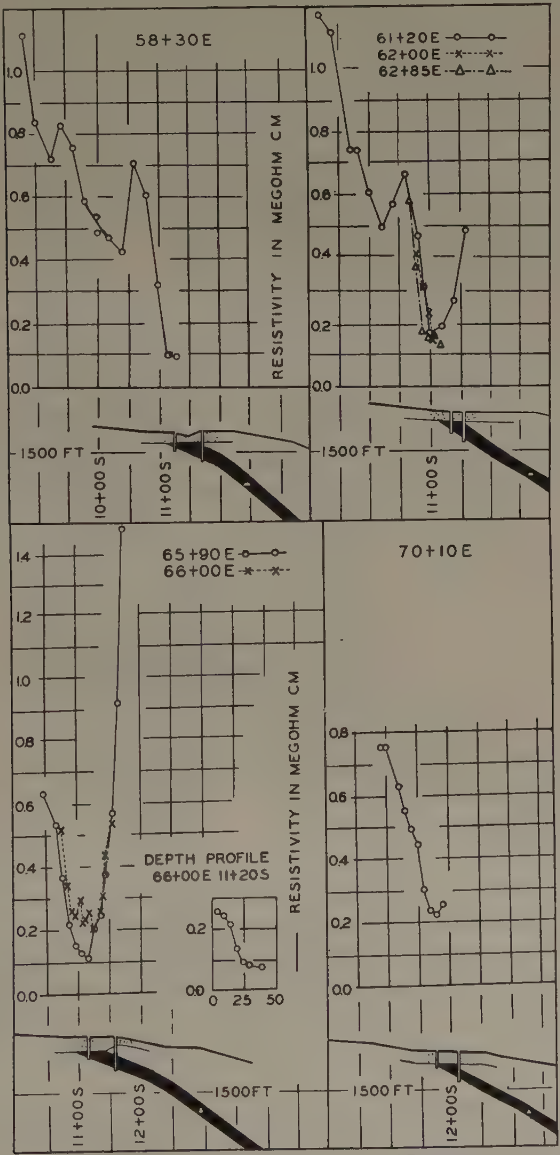

FIG. 14.—SECTIONS FOR PROJECT 3. NORTH AT LEFT.

The traverses on 62 + 85 E. and 62 + 00 E. could not be extended across the crop because of large mine breaches, but these profiles show the approximate crop of the bottom slate of the seam. The profile along 62 + 85 E. is projected to the cross section 62 + 00 E., which explains the slight discrepancy between the two.

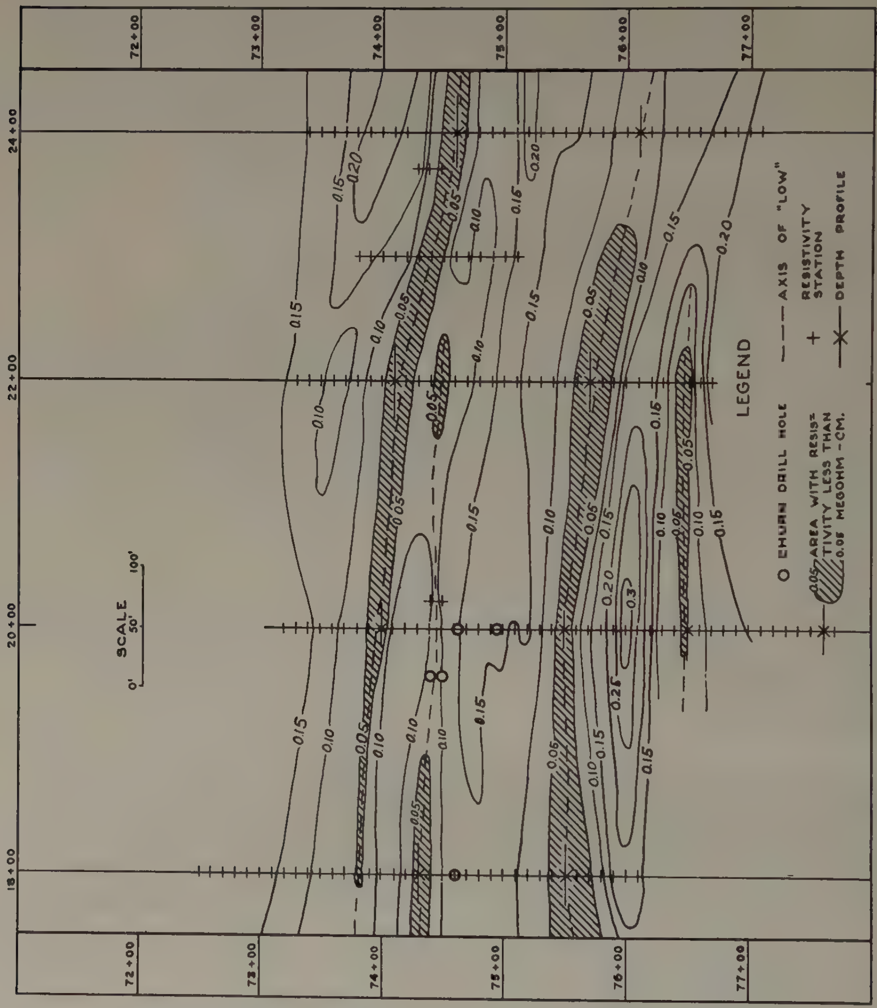

FIG. 15.—RESISTIVITY MAP OF PROJECT 4. NORTH AT TOP OF MAP. Electrode spacing 20 ft. E.-W. Resistivity in megohm-centimeters.

The profile along 61 + 20 E. is also projected to the 62 + 00 E. cross section (Fig. 13). The minimum is a little less than 0.1 megohm-cm., probably because the mantle is thinner here.

This survey demonstrated that the crop of a seam could be located fairly accurately by the earth resistivity method even where the wash was about 20 ft. thick. The high resistivity of the rock above and below the seam is worthy of note.

Project No. 4

The purpose of this project was to locate the crop of the Orchard seam on the south side of the Mahanoy City syncline on several section lines. A few holes were drilled in this area about a year before the electrical survey was made but none have been drilled since the survey was made (Fig. 15). In view of the fact that a simple synclinal structure had been generally assumed, some surprising results were obtained. On all the section lines except 24 + 00 E., the profiles have more minima than can be explained by the simple structure originally assumed. One interpretation of these profiles has been sketched on the cross sections on Fig. 16.

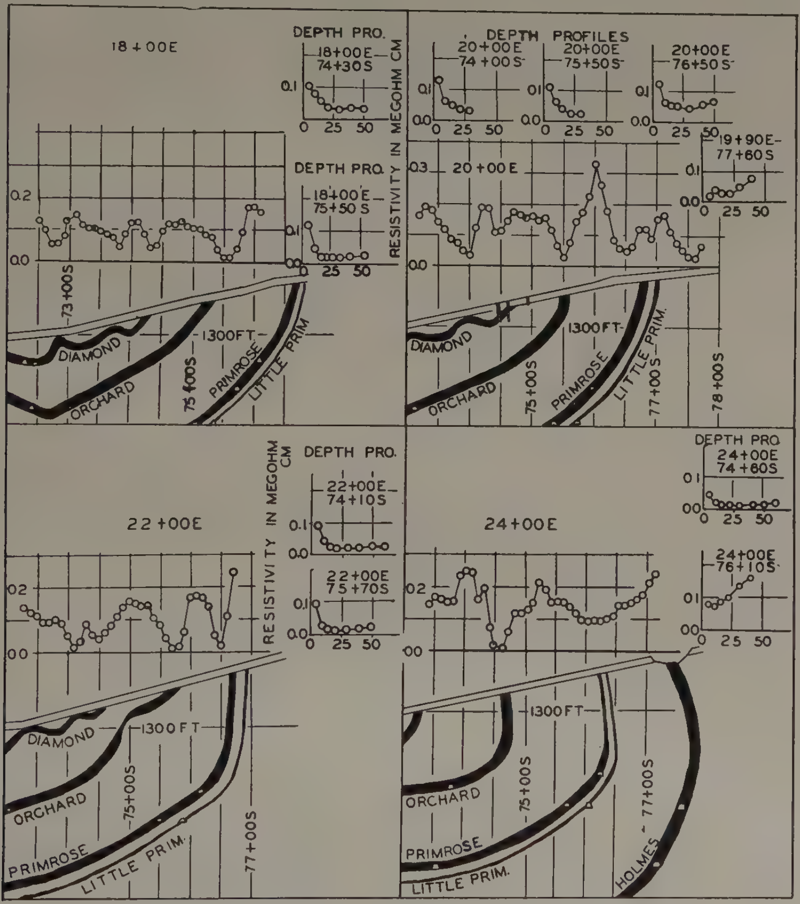

FIG. 16.—SECTIONS FOR PROJECT 4. NORTH AT LEFT.

The profile along line 24 + 00 E. shows a marked minimum as low as 0.01 megohm-cm. at the crop of the Orchard seam. A broad minimum down to 0.1 megohm-cm. occurs where the Primrose should crop but the depth profile is interpreted to indicate that no seam of any size is present.

The names of the seams on the resistivity map (Fig. 15) do not agree with those on the cross section along line 24 + 00 E. It is difficult to see how the resistivity map can be drawn to agree with the cross section. The explanation is not apparent, but it may be that the seams are not correctly correlated on the cross section.

The profile along 22 + 00 E. (Fig. 16) is quite different. Four minima are present. A very sharp minimum was found about 20 ft. north of the point where the Primrose seam was thought to crop. The section must be in error by this amount. The minimum 75 ft. north of the first probably corresponds to the Orchard seam. The Orchard was mined to the pinch shown. The very low resistivity obtained seems to indicate a good thick vein. Two minima were found south of the mine breaches, which were supposed to mark the crop of the Diamond seam. Two small synclines as drawn would explain these.

Except at its south end, the profile along 20 + 00 E. is similar to that just described. It is not clear what the minimum at 77 + 60 S. means but the depth profile shows that the low-resistivity material is near the surface. If a coal seam is present it probably pinches at a depth of 40 or 50 feet.

The minimum at 76 + 50 S. is thought to represent the crop of the Primrose seam. The crop of the Orchard seam is indicated by a sharp minimum. Again two small synclines are needed to explain two minima between the Orchard crop and the breaches of the Diamond seam. A drill hole has shown the presence of coal beneath the smaller of these two minima, which offers some support to this interpretation.

The profile along 18 + 00 E. is similar to those first described. The crop of the Orchard seam is indicated by a marked minimum and three minima north of this are thought to indicate the crop and two anticlines of the Diamond seam. The three profiles 22 + 00 E., 20 + 00 E. and 18 + 00 E. present a consistent picture whatever the interpretation. The small folds in the Diamond seem to offer a more likely explanation than repetition of the Orchard and Diamond seams by thrust faulting.

No resistivity values of more than 0.33 megohm-cm. were recorded on this survey, in marked contrast to the high values obtained in Projects 1, 2 and 3. The rock is largely shale. The drainage conditions were good, which makes the low values even more surprising.

Project No. 5

The location of project No. 5 is about 1½ miles northwest of Girardville, Pa., as shown on Fig. 1. The purpose was to locate the crop of the Buck Mountain seam on several section lines. The standard equipment

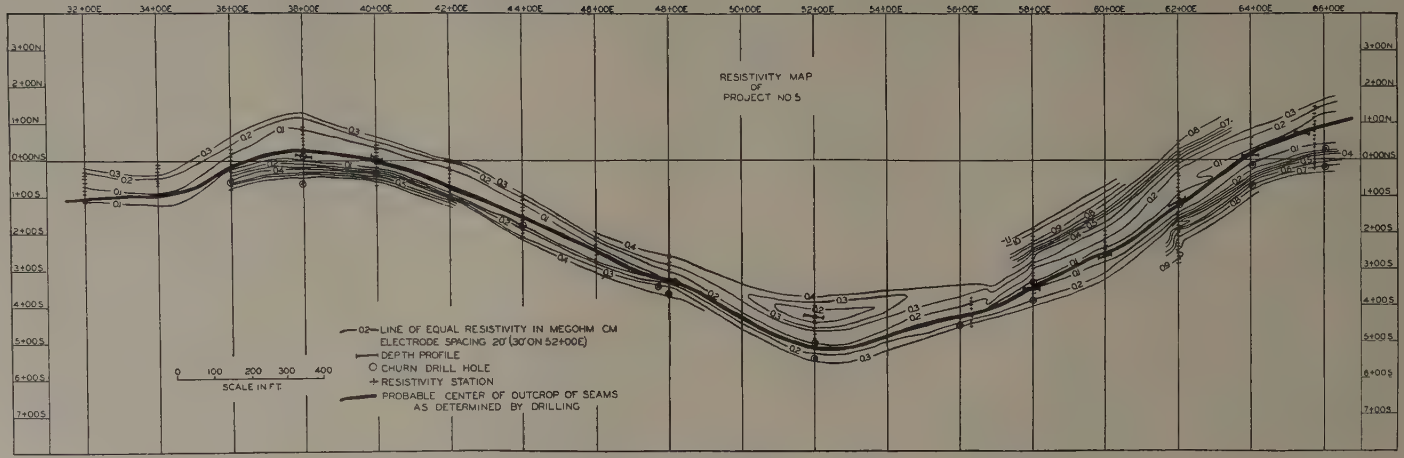

FIG. 17.—RESISTIVITY MAP OF PROJECT 5. NORTH AT TOP OF MAP.

was not used on this survey because the ground was so rough that an automobile could not reach any section line. Since portability was an important consideration, four radio "B" batteries were substituted for the motor-generator set for power. Since the commutator was not used, it was necessary to read the natural potential at each station and to make a suitable correction.

Sixteen transverse traverse profiles and 10 depth profiles were run. A resistivity map (Fig. 17) was constructed from these data. Some of the sections have been drilled, others have not. On all lines where the cross section has been proved by drilling, the agreement with the resistivity profile is very good; in fact, the difference is no greater than the probable error in making the sections from drill records. On sections not drilled, the agreement is not always so close but the inference is that the original cross sections on those lines are wrong.

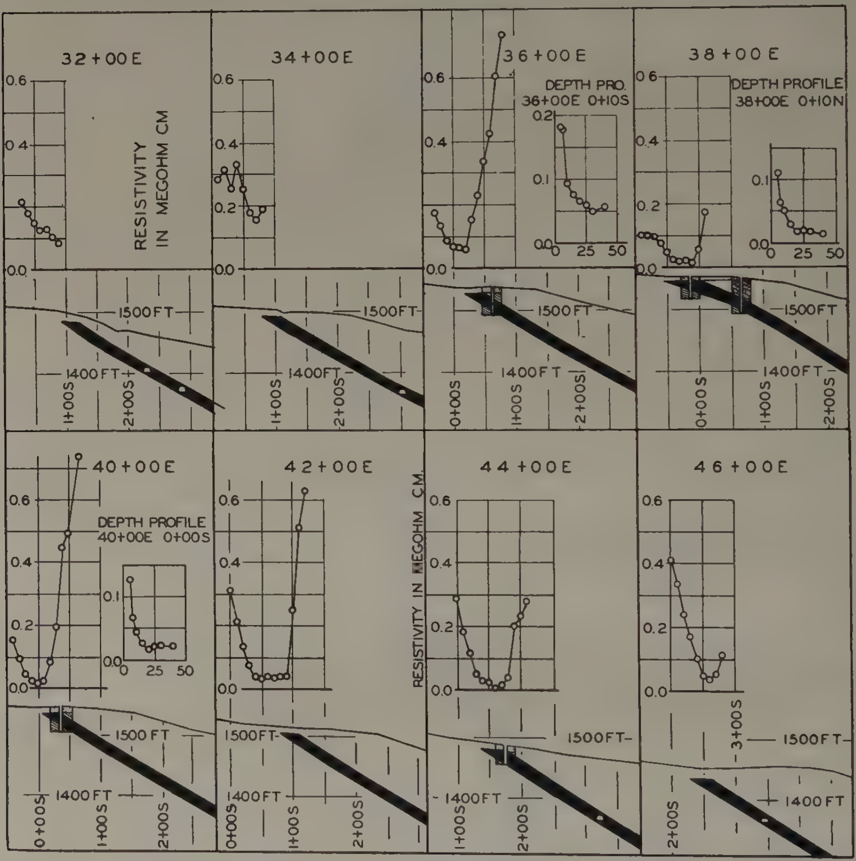

FIG. 18.—SECTIONS FOR PROJECT 5. NORTH AT LEFT.

The variation in thickness of mantle finds expression in these profiles (Figs. 18 and 19). On all lines except 48 + 00 E., 52 + 00 E. and

56 + 30 E. the minimum was less than 0.1 megohm-cm. The higher resistivity at the crop is attributed to deeper mantle rock, and this interpretation is confirmed by the drill records. The lowest resistivity was obtained on the line 44 + 00 E., where a drill hole has shown the mantle material to be less than 10 ft. thick. The depth profiles also aid in interpreting the depth of mantle material, but attempts to solve this problem by Roman's method[11] have been unsuccessful, probably because it is not a two-layer problem. From an observation either of the depth profiles or of the traverse profiles the approximate thickness of the mantle can be predicted.

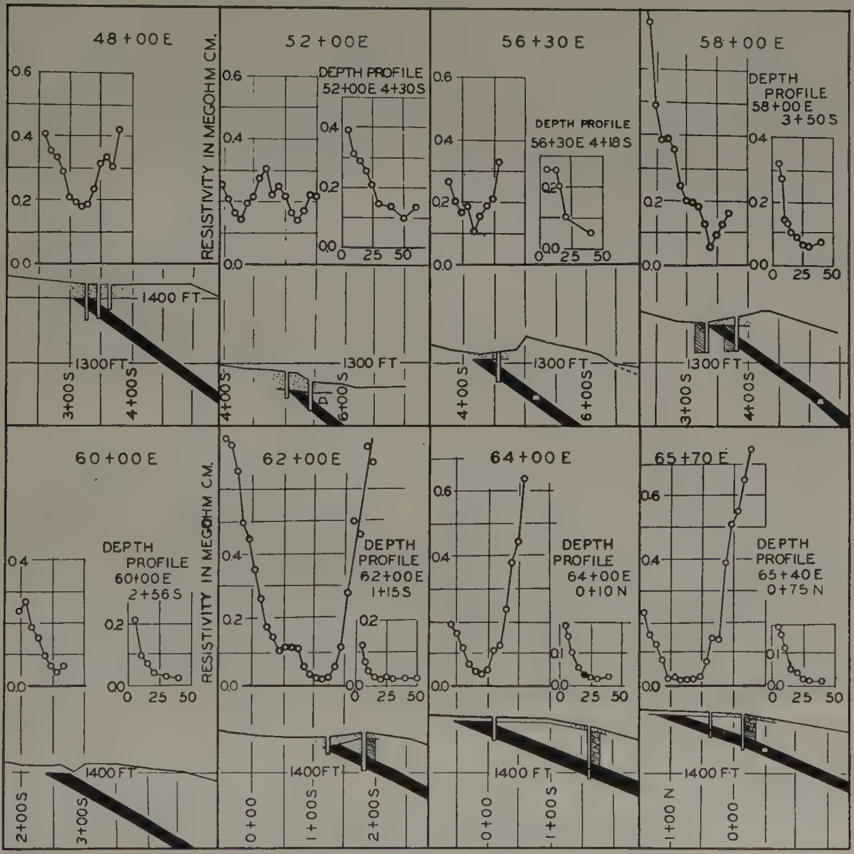

FIG. 19.—SECTIONS FOR PROJECT 5. NORTH AT LEFT.

The transverse traverse profiles and cross sections show the correlation between the numerical value of the minimum and the thickness of mantle above the crop of the seam. The transverse traverse profile yields no information in regard to the thickness of the mantle except above the crop of the coal seam.

The profile along line 52 + 00 E. is different from all the others in that it has two minima, each at about 0.15 megohm-cm. The profile offers no reason why they should be similar and yet the cropping of two seams seems unlikely. The minimum at 5 + 20 S. has been drilled and the seams located. The minimum at 4 + 30 S. is very similar but if it also indicates the crop of a seam the Buck Mountain must be repeated by faulting. If this were true, some evidence of it should be found on other section lines. This is the only profile with two minima. The topography and drainage conditions offer another alternative. The mantle is thick and there is reason to believe there is very good flow of ground water in the vicinity. It is possible that the minimum is due to a saturated zone of the mantle but if so it is surprising that the minimum is so sharp.

The resistivity map (Fig. 17) shows correlation of the crop of the seam with the low-resistivity areas. The Buck Mountain is a very good seam to study in this manner because it is overlain and underlain by sandstone and conglomerate, both of which have very high resistivity.

VARIATION IN RESISTIVITY OF ANTHRACITE

When the use of electrical methods in the anthracite region was considered and information was sought on the specific resistance of anthracite, the data found were scanty and in part at least seemed to be contradictory. The specific resistance of anthracite coal is given in a table by Ambronn[12] as 10^{17} ohm-cm. No other substance in the table has a higher resistivity. Mica is nearest with a specific resistance of 9×10^{15} ohm-cm. followed by quartz with a specific resistance of 10^{14} ohm-cm. It is believed that the value of 10^{17} for anthracite must be a misprint since no other determinations that have been discovered seem to agree with it. Sinkinson measured the resistance of two cubes of anthracite, each 1 cm. on a side. The first gave values of 789 and 4509 ohm-cm. in different directions and the second 992 and 5090 ohm-cm. respectively[13]. Koenigsberger measured the resistivity of coal in mines in Silesia[14]. A solid coal seam was found to have specific resistance of 1.6×10^4 ohm-cm. while a seam with fissures had a specific resistance of 5.7×10^4 ohm-cm., but seams high in CO_2 had unexpectedly high values, greater than 10^7 ohm-cm. The composition of the coal is not given. Schlumberger and Leonardon have described a method of locating coal horizons by resistivity determinations in a drill hole. The coal seam has higher resistivity (about 1000 meter-ohms), than the shales above and below (Ref. 15, pp. 285–287). The composition of the coal is not stated.

Resistivity studies of lignite have been published[16] but these have little bearing on the anthracite problem. The resistivity of powdered samples of anthracite has been measured by Sinkinson[13] and by Meyer[17].

Such determinations are of little direct value to the geophysicist, who must measure the resistivity of material in place, but they are useful in studying the variations in resistivity of anthracite. The results of Sinkinson and of Meyer show a considerable variation in the resistivity of powdered anthracite from various seams and different localities. Though neither investigator directs attention to it, their data seem to warrant the conclusion that the variation follows the same geographic variations shown by Turner[18] for the volatile matter, specific gravity and fixed carbon content. (See Fig. 1.)

Since more information on the resistivity of the materials to be studied in the field was needed, specimens of sandstone, shale or slate, and coal were secured and the resistivity determined in the Physics Laboratory of Lehigh University. The primary purpose was to determine whether the resistivity method could be expected to be applicable to all parts of the western middle anthracite field. Anthracite from other fields was not tested but the resistivity of three specimens of bituminous coal from New River, W. Va., was determined.

Two methods of determining the resistivity of specimens in the laboratory were used. The first was the exact counterpart of the method used for field measurements. Four point electrodes were held in contact with a flat surface of the specimen by means of springs. Current was passed through the specimen between the two outer electrodes and the potential difference between the two inner ones was measured by use of a potentiometer. No commutators were needed. The electrode spacing was usually about 2 cm. This method was convenient and reasonably accurate. Its use was restricted to large specimens having one fairly flat surface. It could not be used for the specimens having very high resistance, because of the difficulty in getting a measurable current to flow between the point electrodes for these specimens.

In the second method the specimen to be measured was shaped into a cylinder or prism usually 5 to 10 cm. long and 1 to 2 cm. thick. An electric current was passed through the specimen in the direction of its long axis. Contact for leading the current into the specimen was made at each end by clamping lead foil tightly against each end. For the specimens of high specific resistance it was very helpful to coat the end with a solution of colloidal graphite* before clamping the lead foil into position. The possibility of error arising from high contact resistance at the current electrodes was avoided by measuring the potential drop between various points along the specimen and not the total applied potential.

Table 2 shows that the specific resistance of all specimens from projects 1, 2, 3 and 4 averages 1.2×10^{-4} megohm-cm. and the variation is not great, being from 0.7×10^{-4} to 2×10^{-4} megohm-cm. No specimens from project 5 were collected but specimens from project 6, near Locust Gap (Fig. 1), average 0.21 megohm-cm. Specimens from project 7, near Shamokin, average 200 megohm-cm. in specific resistance.

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* Aquadag, made by Acheson Colloids Corporation.

TABLE 2.—*Resistivity of Specimens of Anthracite*

Specimen No.	Seam	Locality	Resistivity, Megohm-cm.
KN25	Skidmore	Project 2	0.7×10^{-4}
KN22	Bottom Split	Project 2	1.0×10^{-4}
KN23	Bottom Split	Project 2	1.0×10^{-4}
KN26	Top Split	Near project 2	1.5×10^{-4}
KN27	Top Split	Near project 2	2.0×10^{-4}
KN20	Top Split	New project 2	1.5×10^{-4}
KN50	Top Split	Project 1	0.7×10^{-4}
KN15	Top Split	Project 1	2.0×10^{-4}
KN51	Top Split	Project 1	1.0×10^{-4}
KN12	Holmes	Project 1	1.0×10^{-4}
KN52	Holmes	Project 1	1.0×10^{-4}
KN10	Primrose	Between projects 3 and 4	1.0×10^{-4}
KN18	Orchard	Between projects 3 and 4	1.5×10^{-4}
KN17	Diamond	Between projects 3 and 4	0.5×10^{-4}
KN17	Diamond	Between projects 3 and 4	1.5×10^{-4}
MS34	Mammoth	Near project 6	0.1
MS31	Mammoth	Near project 6	0.2
MS32	Mammoth	Near project 6	0.2
MS33	Mammoth	Near project 6	0.4
A51	Primrose (?)	Project 6	0.3
A52	Primrose (?)	Project 6	0.1
S61	Mammoth	Project 7	100
S62	Mammoth	Project 7	60
S63	Mammoth	Project 7	1
S64	Mammoth	Project 7	600

TABLE 3.—*Resistivity of Specimens of Bituminous Coal from New River, W. Va.*

Specimen No.	Resistivity, Megohm-cm.	Chemical Composition on Dry Basis, Per Cent	
28	60	Volatile matter	26.92
29	150	Fixed carbon	69.46
33	100	Ash	3.62
		Sulfur	0.70

The specific resistance of all specimens studied falls in three groups: (1) A low resistance group including all specimens of the various seams from projects 1, 2, 3 and 4 in the Mahanoy City area; (2) a group of intermediate resistance, including all specimens from project 6, near Locust Gap; (3) a high-resistance group, including all specimens from project 7, near Shamokin. There is a great increase in specific resistance to the west. The map in Fig. 1, which was constructed from Turner's isovol map[15], indicates that the variations in resistivity are concomitant with the variations of volatile content, specific gravity and fixed carbon. Sufficient determinations have not been made to prove this correlation, but the data presented seem to indicate it rather clearly. The resistivity jumps from 1.2×10^{-4} megohm-cm. to 2.00 megohm-cm., with a change of volatile matter from 3.5 per cent to 9.5 per cent, as indicated by Turner's isovols (Fig. 1). This change in specific resistance is amazingly large. Apparently there is not much increase in specific resistance from anthracite of the Shamokin type to bituminous coal, since the resistivity of specimens of bituminous coal from New River, W. Va., with 27 per cent volatile, was found to be about equal to that of the Shamokin anthracite.

Since the specimens of coal studied in the laboratory had dried out and lost their original moisture, it is reasonable to suppose that the resistivity measured in place would be somewhat different. The resistivity of the coal in place was measured several times in open pits and once in a mine breach. At project 1, a good determination was made on solid coal of the Holmes seam, which at this point was 30 to 40 ft. thick. The average of several readings taken in different directions and with different electrode spacing was 3×10^{-4} megohm-cm. This is as close to the laboratory determination (1.2×10^{-4} megohm-cm.) as could be expected.

Near project 5, a measurement of the Mammoth seam gave a resistivity of 3×10^{-3} megohm-cm. At project 6 the field determination was 2×10^{-2}, as compared to the laboratory determination of 2×10^{-1} megohm-cm. At project 7, the conditions for the determination of the resistivity in place were very poor, so the value obtained of 1 megohm-cm. must be taken as only approximate. This is less than the value obtained

TABLE 4.—*Resistivity of Specimens of Rock from the Western Middle Anthracite Field*

Specimen No.	Rock	Horizon and Locality	Resistivity, Megohm-cm.
KN11	Shale	Divider in Holmes seam, project 1	1.5
KN15	Grey micaceous shale	Project 1	6.
KN29	Sandstone	Near project 2	2.
KN19	Gray sandstone	Between projects 3 and 4	10.
S70	Quartzitic sandstone	Project 7	40.
S71	Black shale	Project 7	100.
S72	Black shale	Project 7	200.

in the laboratory; namely, 200 megohm-cm. Except at project 1, the resistivity of the coal in place was lower than that of the corresponding specimens measured in the laboratory. Moisture in the coal in the field probably is the reason for this difference.

Determination of the resistivity of a number of specimens of sandstone and shale was made in order to learn how it would vary from the resistivity of the anthracite. The results are shown in Table 4. The specific resistance of the shale specimens is not very different from that of the sandstone. The resistivity measured in the field above sandstone beds was at many places as high as some of the values obtained in laboratory specimens of sandstone. The resistivity in the field above shale beds was found many times to be as low as 1.5×10^{-1} megohm-cm., compared to a resistivity of laboratory specimens of 1.5 to 200 megohm-cm. It must be remembered that in field determinations of resistivity, the resistivity of the mantle always affects the results. The mantle above sandstone beds is usually stony, porous and without much clay. Such material would be expected to have high resistivity. Also, since the sandstone is resistant to weathering and erosion, the mantle would usually be thinner than it would be above a weaker rock such as shale. The mantle above a shale is usually composed largely of clayey material, which retains the moisture and would be expected to have low resistance. Clayey mantle has been found to be as low as 10^{-1} megohm-cm. Moreover, it seems probable that the water held in shale, which is fine grained and relatively impermeable, may contain more salts in solution than the water in a sandstone (ref. 19, p. 239). However, regardless of the explanation, the fact remains that the resistivity of shale in the field is less than it is in the laboratory, whereas the values for sandstone are about the same.

The success of the projects described in this report depended, of course, upon the marked difference in the resistivity of anthracite as compared to that of the overlying and underlying rocks. The laboratory work has shown greater difference in the resistivity of anthracite itself than between the anthracite from Mahanoy City and sandstone or shale of that locality. The highest resistivity obtained in the laboratory was upon a specimen of anthracite from near Shamokin. It is apparent, therefore, that anthracite beds probably cannot be located by the earth-resistivity method in some places, since there will be not sufficient difference in the resistivity of coal and rock. If it can be assumed that the variation in resistivity is everywhere concomitant with variation in volatile content, a study of the isovol map in Fig. 1 will reveal the localities where the resistivity method would be expected to be applicable. It has been demonstrated that the method can be applied in the western middle field east of Ashland and it seems probable that it can be applied 5 miles west of Ashland. The territory in the southern field east of Tremont, all of the eastern middle field, and southern part of the northern field seem favorable.

If at any place the resistivity of the anthracite is significantly greater than that of the country rock, it may be possible to locate seams by picking maxima instead of minima on the profiles.

The amazing variation in the resistivity of anthracite is probably of sufficient interest to justify further study entirely aside from its connection with electrical prospecting.

SUMMARY

1. Transverse traverse profiles were made across the crops of anthracite seams.
2. The minima on the profiles, provided they were low enough, indicated the crops of seams. A few minima were due to low-resistance mantle. This fact was determined by depth profiles.
3. Most of the results have been checked by subsequent drilling.
4. The correlation between the resistivity profiles and those of the cross sections that have been proved by drilling is very close.
5. Laboratory study of specimens from the western middle anthracite field has shown a variation in the resistivity of anthracite from 10^{-4} megohm-cm. to 200 megohm-cm.
6. The laboratory data suggest that the variation in resistivity is concomitant with the variation in volatile content.
7. The region east of Ashland in the western middle field has been shown to be suitable for resistivity surveys and it is suggested that the area east of Tremont in the southern field, all of the eastern middle field, and the southern part of the northern field are probably favorable for such prospecting.

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DISCUSSION

(*Lewis E. Young presiding*)

H. G. TURNER,* State College, Pa. (written discussion).—Messrs. Ewing, Crary, Peoples and Peoples are to be commended for their excellent report on prospecting for anthracite by the earth-resistivity method. It is believed by the writer, however, that this method of prospecting for anthracite can be improved through a more extensive study of blocks of anthracite in the laboratory. Myer has demonstrated that moisture is an important factor in the electrical resistivity of granular anthracite. Ewing et al. recognize that moisture plays a part in their field measurements but they might be able to evaluate this factor more precisely after a thorough laboratory study.

To a student of coal structure and constitution it is evident that the position of the electrodes relative to the coal laminae is rather important. Most anthracites are made up of alternate laminae of different physical and chemical nature. Extremely thin layers of shale are often found parallel to the other constituents of laminated coal. These inorganic layers often reach considerable thickness. It is obvious that the conductivity across a condenser-like mass of alternating layers of shale and carbon would be quite different from the conductivity parallel to the carbon layers.

It happens that most of the measurements in this study were made across the laminae of tilted anthracite beds. Had they been made along the strike the results would have been quite different and less significant.

It seems to me that some of the unusual variations in resistivity might be explained through a more extensive study of large blocks of anthracite, measuring conductivity in various directions through the block and under different degrees of dryness.

J. G. KOENIGSBERGER, Baden, Germany (written discussion).—This interesting research is based on the difference in electric conductivity of anthracite seams and of the adjacent sandstone and shales. The field work discovered the anthracite outcrop (under a cover of about 5 m.) almost by its much lower resistance. The resistance of the dry specimens tested in the laboratory confirms these results. The anthracite is so "graphitic" that its resistance in the dry state is very low and would not be much lower in the field outcrops; but the resistivity of the adjacent rocks, dry in the laboratory, is much higher than in the field, as the authors have stated for shales. For sandstone these values were about the same. It seems preferable, therefore, to make investigations in the laboratory always on wet samples and to lay stress upon direct resistance measurements so far as field outcrops make it possible.

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S. F. Kelly,* New York, N.Y.—One phase of this work that I should like to emphasize illustrates, in my opinion, the most important application of geophysics; that is, the outlining of geological structure by geophysical observations. On the resistivity maps, not the profiles, but of the areas as a whole, note that the equiresistivity lines gave clear indication of the underlying structures, which otherwise were concealed by the "wash" lying on the bedrock.

Prof. C. Schlumberger, under whom I worked in France and North Africa in 1920–21, was the first to direct attention to the possibility of searching for coal by electrical methods. On my return to the United States in 1921, I carried out a series of experiments in Tennessee, Ontario and Michigan, on metal deposits, and near Wilkes-Barre, Pa., on anthracite beds. The results were published in 1922 (see reference 4). Those early experiments of mine, which marked the inception of geophysical methods on a commercial scale in the United States and Canada, dealt almost exclusively with the self-potential, or spontaneous polarization method. For the benefit of those who are not geophysicists, I shall explain this briefly, because, in spite of the skepticism of the present authors, it remains as a possible technique in the rapid prospecting for anthracite.

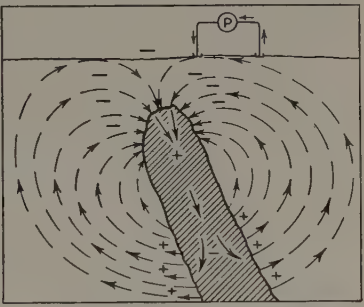

Fig. 20.—Schematic cross section of flow of currents spontaneously generated by metallically conductive bodies.

Including diagram of current path through measuring apparatus at surface. *P* is a potentiometer.

Sulfide ores possess a metallic conductivity for electricity, whereas most other rocks and minerals owe what electrical conductivity they show to the moisture in their pores, and to the salts dissolved in this moisture; in other words, they are electrolytic conductors. A metallic conductor, such as a sulfide vein, that extends near the surface has its upper part in a region where the moisture of the adjacent rocks is oxidizing, owing to oxygen brought by rain water and acid from the weathering of the sulfide minerals. Deeper within the earth the moisture is neutral, or even reducing in character. Thus, such a body fulfills the requirements of a galvanic cell—a metallic conductor joining two dissimilar electrolytes—and a current flows down the metallically conducting body, out into the country rock, and back up to the surface, where it completes the circuit by flowing in to the apex of the vein (Fig. 20).

Such a body may be prospected for by making systematic observations of the strength of the ground currents from place to place, and noting the areas where there seem to be centers of electrical activity. The voltages measured will usually be of the order of a few hundredths to a few tenths of a volt. This technique is applicable wherever a mineral body of metallic conductivity extends above and below the water table, and hence is exposed to oxidizing action on its upper portion.

Evidently the question resolves itself into this: Is anthracite a metallic conductor? Had the present authors been more familiar with the answer, they probably would have refrained from describing the results of my experimentation as being almost entirely due to stray currents.

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The answer is: Anthracite sometimes is a metallic conductor, and sometimes it is not; usually it is. A simple laboratory test, which Professor Schlumberger taught me in Paris some 15 years ago, will quickly demonstrate which variety is under investigation. Thin seams of black graphitic material may be seen in a lump of anthracite. If a pair of wires is connected to a battery in series with a galvanometer, and the bare ends of the wires are touched to the lump of anthracite, at the point at which they both make contact with the black, sooty seams the galvanometer needle will show a marked deflection. This indicates a high conductivity along the seam. But, if contact is made off the seams, this is *not* so.

The question naturally arises, what is this material that endows some coals with such a remarkable conductivity? Having just returned from Canada, and having had no time to make a thorough search in recent literature, I shall refer in this connection to an old paper, "The Microscopical Structure of Anthracite," by H. G. Turner, published in 1925 (A.I.M.E. pamphlet), with which I am familiar through having been interested in studying the microstructure of coal myself. At that time (1925) I was comparing anthracite and anthraxolite microstructures, so unfortunately did not study this question of the sooty seams in anthracite. Later authors may have published more extensive research. However, I shall quote from Turner, as follows:

"Mineral charcoal is abundant in almost all anthracite. Cleaving coal parallel to the banding exposes charcoal-covered surfaces in every specimen that shows distinct lamination. In fact, it is almost impossible to find a square inch of laminated coal that does not show charcoal either in thin sheets or lenses. On the other hand, the varieties that are poorly laminated and almost uniform in luster contain little charcoal. The absence of distinct lamination is no doubt due, in part, to the paucity of dull charcoal. In addition to the thin sheets of charcoal so commonly found in the bedding planes, one finds occasional charcoal lenses several inches in thickness composed of crisscrossing chips two centimeters or more in length and one centimeter in width. Detached chips of these dimensions are also encountered with the coal bands warping around them."

Probably the bands he refers to are the black, sooty bands that endow the coal with metallic conductivity, so that it can generate a current to be detected at the surface of the ground. Turner further says:

"The carbonized matter of coal has been called 'charcoal,' 'mother of coal,' and 'fusain.' That there is a great amount in anthracite is indicated by the foregoing megascopic description. This material varies in hardness and texture and, possibly, in composition. Most of it shows good plant structure, being structureless in rare cases only. Figs. 18* and 19 represent cross sections of large fragments of material that, to the unaided eye, look like fragments of ordinary charcoal. One piece (Fig. 18) is so soft that it crumbles readily in the fingers: the thin walls of the perfect cells show the reason for this behavior. The other piece (Fig. 19) is so hard that it can just be scratched with the finger nail. The reason for this superior hardness is shown in the much thicker cell walls and component bright bands of hard coal material. In both of these fragments the cells are devoid of filling. Fragments of other charcoal-like material were isolated and found to be too hard to be scratched with the nail. Under the microscope, these hard fragments show crushed and fractured cells, as illustrated in Fig. 5. It is suggested that this particular material represents carbonized wood which had been densified by infiltration of products of vegetable decay and further hardened by compression.

"The charcoal-like materials are similar in that they show almost perfect structures. This would suggest that they escaped the jelly stage of the other materials in the coal. Difference in thickness of cell walls and of the cell filling may indicate differences either in stages of carbonization or oxidation before burial, or in original composition or in behavior after burial."

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* The figure numbers in the quoted material refer to figures in Turner's paper.

The megascopic description of the coal by Turner gives evidence that these conductive seams may, on occasion, be absent. Under such circumstances the coal would not be metallically conductive, and would not give self-potential currents. Thus we see that it is perfectly possible that I observed such currents at Franklin colliery, while Ewing and his collaborators failed to find them at another location.

Let us see the results of my traverse at Franklin colliery: At the time the readings shown in Fig. 21 were made, we were all green in geophysics, and I published this chart without making some needed corrections, and it is on that account that Mr. Ewing feels that the correspondence with underlying structure is poor. There was apparently an electric unbalance between the electrodes I was using, which caused the readings to go up continuously, obscuring the true shape of the profile. Fig. 22 shows a corrected profile which I drew for the discussion, which shows that the peaks of electric activity correspond with the positions of the coal veins. The Skidmore vein gave a marked peak, whereas a thin seam lying very flat, the Baltimore vein, probably with weaker electric activity, merely produced a hump on the decline from the Skidmore vein. I was in the same situation there with respect to the coal section that Mr. Ewing and his coauthors were in; namely, the exact position of the vein was not known at the spot where I worked.

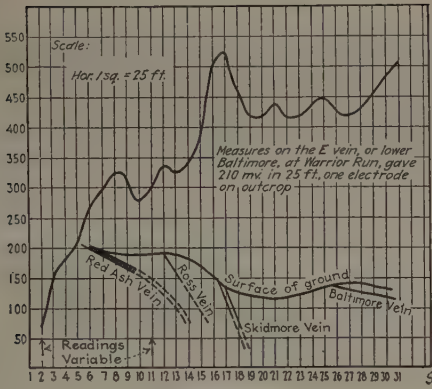

FIG. 21.

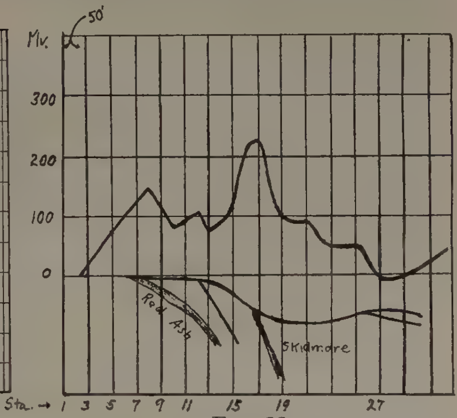

FIG. 22.

FIG. 21.—RESULTS OF FIELD WORK AT FRANKLIN COLLIERY OF THE LEHIGH VALLEY COAL CO. (RED ASH, ROSS, SKIDMORE AND BALTIMORE COAL SEAMS).
FIG. 22.—CORRECTED PROFILE AT THE FRANKLIN COLLIERY.

The spontaneous polarization method has been successfully used to prospect anthracites in the Alps, for example, where Professor Schlumberger and his collaborators found that the coal suboutcrops were clearly indicated by regions of intense electrical activity.

Turning to the difficulties the present authors encountered in interpreting their depth profiles, I would warn them that the ideal two-layer case is almost nonexistent. They used the two-layer method of interpretation, and it has been my experience that this ideal case almost never exists in nature. As soon as more layers enter the picture, the interpretation becomes more difficult.

We recently did some work in Quebec where I thought we had a two-layer problem. We obtained apparently typical two-layer curves, but when we came to interpret them, we found a weathered layer, then the unweathered glacial drift, and beneath that the granite bedrock, so that they formed three layers of increasing resistance with depth. The result was that the curves took the typical two-layer shape, although there were actually three layers, and the interpretation was accordingly much complicated.

Another point about depth determination is that one seldom finds the ideal case of homogeneous media, in either overburden or in subadjacent bedrock. In depth determination in the coal field, the underlying bedrock is anisotropic. There are dipping strata here, and one may be sandstone, another shale, another may be coal. Consequently, the subadjacent material does not have uniform resistivity; also, the resistivity parallel to the strike will be quite different from the resistivity perpendicular thereto. A depth profile in which the measurements are made parallel to the stratification will give entirely different results from a depth profile in which the measurements are made perpendicular to it. It is even possible that in the one case the resistivity of the bedrock will be greater than that of the overburden, and less in the other; and at just the right angle to the underlying stratification one might find the resistivities equal!

So it becomes necessary to know the direction of the electric measurements with respect to the stratification. Knowledge of direction of readings is essential for an appreciation of the value of the depth profile and for a valid interpretation. The question of anisotropy needs to be studied carefully.

One other point, which undoubtedly has been taken into account by Mr. Ewing and his coworkers but which I make merely to correct what possibly might be a misapprehension on the part of some one: Mr. Ewing mentioned the fact that the resistivity profiles were better when made as transverse traverse profiles. That is, the four electrodes are arranged in a line parallel to the strike, and are moved perpendicular to their own line in the successive measurements. But that is not always the best technique. It is best for picking up a conductive layer, such as the coal, but if one is following a resistant stratum, an arrangement of the electrodes perpendicular to the strike is the best one, because the stratum acts as a wall perpendicular to the line of current. If the electrodes are parallel to this wall, there is little effect from one electrode to the other, because the current does not have to go through that resistant wall.

The work these men have done is extremely valuable for showing the practical methods of studying structure, and of directly locating mineral bodies, and of these the study of structure is undoubtedly the most valuable.

In this connection, I should like to direct attention to a remarkable piece of work that was carried out in Spain a few years ago; the paper describing it, by de la Peña and Sineriz, was presented before the A.I.M.E. geophysics group last February, and is available in manuscript form in the library. Briefly, the Spanish Government was anxious to find, on Government-owned property, the extensions of some coal seams being mined on adjacent, privately held land. Seventeen drill holes had failed to find a single seam of coal! Finally, geophysical surveys were decided upon, and a well coordinated program of electrical, magnetic, gravitational and seismic work was undertaken. As a result, a number of faults were discovered and traced, and the fault blocks were outlined. It was furthermore found possible to tell which blocks had been pushed up and eroded and which ones had been dropped down and deeply buried by later sediments. One comparatively undisturbed block was located, and the first drill hole put down on the geophysicists' recommendation encountered five seams of coal at exploitable depth! Fifty million tons of coal are believed to be available in this block.

I bring this in to emphasize the fact that geophysics must be regarded not so much as a means of locating directly a given type of mineral occurrence, but rather as an aid to the geologist in his unraveling of difficult problems of structure, especially where these have a bearing on the question of where to concentrate further exploration for possibly commercial mineral bodies.

(*Sherwin F. Kelly presiding*)

G. H. ASHLEY,* Harrisburg, Pa.—I do not feel inclined to discuss this paper from the technical standpoint, but I am wondering where our work as directors of geological surveys is going to stop! Some years ago we made topographic maps of that region which today are treated as reconnaissance maps. Today we are making, around that region, maps that we feel are 100 to 200 per cent better. In fact, we now have a W.P.A. project on foot to re-map the anthracite region, as soon as we get the money and provided we get the money. But that will be on 7½-minute quadrangles on the scale of 2000 ft. to the inch. That simply gives the thought that is in my mind, of the constant stepping up of the scale and detailed character of the work demanded of us. When the present Pennsylvania survey was established in 1919, the Governor, with whom I was talking it over, said: "I thought the geology of Pennsylvania had all been mapped." Of course, we have a state map and county maps, and so forth, but a comparison of some of those old maps with what we are now getting out, will show the difference that has come about. Years ago we thought we could work with a compass and a barometer; today we are using the spirit level and two men are required where formerly only one was needed. With the increasing amount of state land being taken over and the increasing number of questions that we are asked to settle as to forest preserves, game preserves, and so on, and as to whether there is coal under this or that land, and so forth, I can see that the geological survey's life will be extended for probably another century.

Some of you are familiar with the 6-in. maps of the English survey; and I judge, from the difference in the type of work that survey has done, that the time will come when the surveys in the United States will be working, perhaps, on the same scale, and will look on these maps as necessary tools for work.

J. W. PEOPLES (written discussion).—The value of the earth-resistivity method of locating "crops" of anthracite seams must depend upon the need of the information. Where the mine maps are sufficiently complete and detailed, obviously electrical prospecting is not needed to locate "crops." At other places, because of surface conditions, surveys may not be feasible. Nevertheless, the results shown seem to me to warrant the belief that the method can be profitably used by the mining companies provided judgment is used in selecting conditions suitable, in making surveys and in interpreting the profiles.

The surveys described by us were made in a region that had been mined for many years. Furthermore, a great deal of prospecting by drilling had been done in the region before the electrical prospecting was attempted. In projects 2 and 4, drilling had been abandoned because of the large number of barren holes in comparison with the amount of coal located. In project 2, after the electrical survey was made on three additional lines, the seams were drilled with only three barren holes on those lines. The stripping area was extended for 600 ft. because of this. Project 4 has not been drilled since the electrical survey was made, but it is believed that the "crops" of the Diamond and Orchard seams have been located. Whether or not there is a stripping can best be determined by drilling.

Project 1 was recognized as an excellent test for the resistivity method, since information was badly needed, inasmuch as the region had been little mined and few data were available for making cross sections. The original cross sections, which were the best that could be made from the information available, were found to be considerably in error, as shown by the electrical prospecting and subsequent drilling, and in some cases by stripping. Fig. 23 shows the outcrops of the various seams as shown on the original cross sections and as determined by later drilling based upon the electrical survey. It is obvious that the location of these seams by drilling without the aid of electrical prospecting would have been costly. That this prospecting was worth while is shown by the strippings located to date.

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* State Geologist of Pennsylvania.

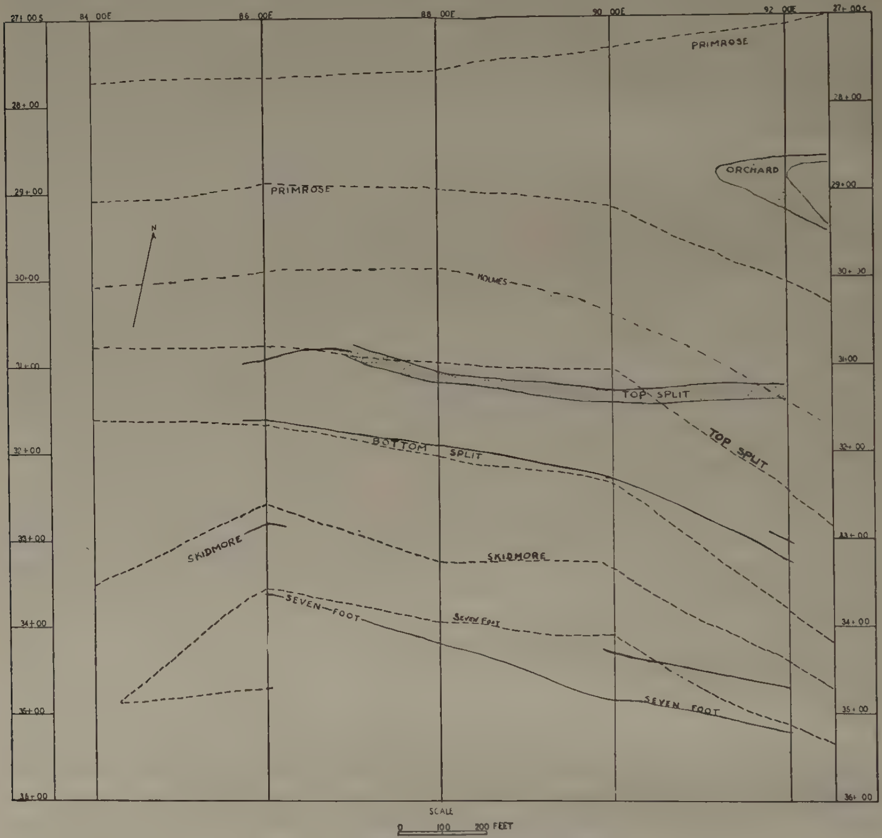

FIG. 23.—SKETCH MAP OF PROJECT 1.

Broken lines indicate "crops" of the various seams as taken from the original cross sections. Stippled areas show actual "crops" of the Orchard and Top Split seams where they have been stripped. Where subsequent data are available the "crops" of the other seams according to this latest information are shown.

The Orchard syncline shown in Fig. 5 was not on the original cross section. A small stripping resulted after electrical prospecting had indicated coal here and subsequent drilling had proved it. Another company followed this same syncline for several hundred feet along the strike and stripped it. A stripping on the Top Split yielded several thousand tons of coal. The original survey of project 1 required only five days, therefore the cost of obtaining this information was low.

M. EWING (written discussion).—A brief test of the equipotential method was made at the start of our work. The results were not promising. Tests were not carried far enough to justify publication of the results. Electrical measurements did not supplant churn drilling. The proper function of the electrical survey in the present case was to offer guidance in the selection of drilling sites, greatly reducing the amount of drilling necessary to yield the information needed.

A subsequent resistivity survey, made and interpreted by inexperienced operators, ended in failure. No geophysicist will be surprised at this result. Operators in the petroleum industry have had enough contact with geophysical methods to realize fully that a broad background of experience in obtaining and interpreting geophysical data are essential if discouraging errors are to be avoided.

Mr. Turner and Mr. Kelly refer to banding and other small-scale variations in the structure of anthracite as possibly producing large differences in resistivity in different directions. In our laboratory measurements of resistivity the specimens were often in the shape of parallelopipeds, several centimeters on the side, and the resistivity in three mutually perpendicular directions was measured. The ratio of highest to lowest value obtained for a given specimen was rarely as large as 2. Of course, some care was exercised in selecting specimens that appeared to be fairly homogeneous.

In connection with Mr. Turner's remarks about the direction of flow of current in the field measurements, it should be pointed out that the lines of flow between two point electrodes placed at the surface of the earth constitute a family of curves, roughly elliptical in shape, which have a variety of directions. In the greater part of our work the line of electrodes, and hence the general directions of flow of current, was parallel to the strike of the structure.

Mr. Kelly's expressed opinion on the difficulties in computing depths from depth profiles is in agreement with ours. Our only serious use of depth profiles was to infer, from the general shape of the curve, whether a given region of low resistivity was due to a coal seam extending to considerable depth or to surface drainage conditions.

Pennsylvanian Coals of the Southeastern Margin of the Western Interior Province

By C. M. Young,* Member A.I.M.E.

(St. Louis Meeting, October, 1935)

This is an attempt to bring together some of the knowledge of the coal-forming conditions obtaining during the Pennsylvanian period in the Western Interior Coal Province, to sketch briefly the present extent and character of the coals produced, and to show the beginning of correlation of beds over wide areas. Coals of later age than the Pennsylvanian are not included and, for reasons that will appear later, little attempt is made to cover the districts of eastern Oklahoma and western Arkansas, and northern Texas is omitted entirely.

Geological Conditions

The conditions that made possible the formation of large coal beds in the United States were most favorable in the Pennsylvanian period, formerly known as the Upper Carboniferous. The geological conditions can be determined from the records, but not the reasons for them. Preceding the Pennsylvanian was the Mississippian, formerly known as the lower Carboniferous and still later as the sub-Carboniferous. During this time, large bodies of limestone were deposited and at the close of the Mississippian they were warped upward and for a long period were eroded. Then there began a period of subsidence, more or less interrupted at times, during which the Pennsylvanian strata were deposited (see table on next page). These consist of sandstones, limestones and shales, with occasional coal beds.

At the beginning of the Pennsylvanian there was subsidence, which allowed the sea to extend over the eroded Mississippian surface to the north and east. It passed over eastern and central Texas, a large part of Oklahoma, with a projection into Arkansas, over Kansas and part of Nebraska, all the northern and western part of Missouri, a large part of Iowa except a strip in the north, most of Illinois, practically all of Kentucky, northern Tennessee, most of Indiana, all of the eastern part of Ohio, practically all of West Virginia, western Maryland, the west end of Virginia, and a large part of Pennsylvania. The Ozarks were already up and separated the main body of this sea, which covered the northern

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and western part of Missouri, from the projection into Arkansas. Also, the Cincinnati Arch was already in position, and separated from the main area of subsidence an arm that projected across northwest Indiana into Michigan. Pennsylvanian rocks are also found extending to the Pacific, but are not considered here.

Separated into various more or less disconnected coal districts, as this great area is now, it has become apparent that all of these fields are parts of one great district in origin. This statement is not to be interpreted as meaning that single coal deposits were continuous over this area, but that coal-forming conditions existed over practically all of it at one time or another. It has been found, however, that the areal extent of some beds is truly remarkable, though we have no proof as yet that these beds are always uninterrupted.

TABLE OF PENNSYLVANIAN FORMATIONS

Appalachian region	Kansas—Nebraska		Illinois
Monongahela	Virgil	Wabaunsee Shawnee Douglas	(Absent) ?
Conemaugh	Missouri	Pedee Lansing Kansas City Bronson Bourbon	McLeansboro
		Greatest spread of Pennsylvanian seas	
Allegheny	Des Moines	Marmaton	Carbondale
		Cherokee	
Pottsville			Pottsville
(Absent)		(Absent)	(Absent)

This table is reprinted by permission from A Textbook of Geology, pt. II, 3d Ed., by Charles Schuchert and C. O. Dunbar. Published by John Wiley & Sons, Inc., New York, 1933.

This subsidence began in the southwest and reached its greatest extent there. For this reason, the sediments of Pennsylvanian age are very much thicker there than in the north and east, except that the extension into Pennsylvania shows great thickness. In most places there are at least 3000 ft. of Pennsylvanian strata, about 9000 ft. in Alabama, and about 16,000, or possibly more, in Oklahoma.

At the beginning of this period the portion of the United States that was then above sea level was a great lowland, bordered on the south and east by a long mountain chain. The character of the sediments deposited in this inland extension of the sea indicates that the bordering highlands were uplifted at various times and thus were enabled to keep up a more or less continuous supply of sediments.

As the basins subsided, filling kept approximate pace with the subsidence. Under these conditions a rather small movement could change the character of the material deposited, making it sometimes sand, sometimes shale, and sometimes limestone.

CORRELATION BY CYCLOTHEMS

A most interesting characteristic of this Pennsylvanian period is the repetition, again and again, of approximately similar conditions, so that we have numerous groups of strata repeated over and over.

The series of rocks making up one of these groups, called a cyclothem, is listed as follows by Wanless and Weller[1]:

Marine sediments:
8. Shale with "ironstone" nodules and bands.
7. Limestone with marine fossils.
6. Black sheety shale with large concretions.

Continental sediments:
5. Coal.
4. Underclay.
3. Limestone without marine fossils.
2. Sandy shale.
1. Sandstone unconformable on lower beds.

The occurrence of such groups of rocks indicates subsidence accompanied by the deposition of sandstone shale and limestone. The following of this subsidence by immediate re-elevation may not be certain, but seems probable. Whatever the cause, the limestone was followed by clay and coal. Then another subsidence caused the coal to be covered with shale, limestone and shale. A final elevation permitted erosion and closed the cyclothem.

Those interested in coal have made many attempts to correlate beds and, where possible, to trace beds from place to place. Sometimes mistaken identifications have been made because two beds seemed to be at about the same horizon. We have sometimes said that a bed changed its top or bottom and sometimes that the character of the bed changed. Sometimes it did change with this change of enclosing rock, but sometimes it was a different bed.

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[1] H. R. Wanless and J. M. Weller: Correlation and Extent of Pennsylvanian Cyclothems. *Bull.* Geol. Soc. Amer. (1932) **43**, 1004.

This recent work on the repetitive nature of Pennsylvanian sedimentation has made it possible to make some positive identifications and some corrections that apparently could not have been made without it.

Also it gives a new conception of the areal magnitude of some of the beds and lends a new interest to the study of them and of their enclosing rocks. Referring to the work that has been accomplished, Wanless and Weller say[2]:

> All of these studies have shown (1) that the entire Pennsylvanian system in the Eastern Interior and northern Appalachian basins and the lower Pennsylvanian strata in the northern part of the Western Interior basin consist of a similar succession of cyclothems, (2) that individual cyclothems are persistent, and (3) that correlation of cyclothems at widely separated localities is possible.

These studies have given confirmation to the idea that there was more continuity than was commonly recognized through the coal fields of this Western Interior Province; also, to the idea that the separation of the Western Interior Province from the Eastern Interior Province along the line of the Mississippi River was an accident and that it might well be possible to trace, if not individual beds, at least beds occupying nearly the same horizon, through all of this great series of deposits. If this proves possible, there is no reason to stop but the work may be continued until there is reasonably close correlation from eastern Pennsylvania to Texas. This will require an immense amount of work.

Definite correlations have been made, including some corrections in Illinois; some of the cyclothems of Illinois can be traced into Iowa and their coals definitely identified with those of the Illinois-Indiana fields, whether or not the beds were actually ever continuous. Since deposition was generally continuous over southern Iowa, northwest Missouri, eastern Kansas, and northern Oklahoma, some, at least, of these cyclic depositions can be traced from Iowa into northern Oklahoma and perhaps into eastern Oklahoma and Arkansas. It is not yet possible to extend the names of the Illinois coals, or whatever names may later be given to them, far beyond the southeast corner of Kansas, simply because the detailed geological work has not been carried far enough to make definite correlation possible.

Illinois Coals and Cyclothems

It may be well to refer to the geological column of northern Illinois, which is the most familiar of those published, and in which many of the coal beds can be placed definitely. At the bottom, and resting on the pre-Pennsylvanian is a cyclothem known as the Babylon. Above this, but somewhat indefinitely known (and in fact these earlier cyclic deposits are not as distinct as some of the later ones) are one or two without definite names. Then one known as the Pope Creek, and above this the Seville. Though the lower groups contain coal beds, it is not until the Seville is reached that we find the coal No. 1, which was at first supposed to be the lowest coal of the Illinois deposits. Above this come in series the DeLong, the Seahorne, the Wylie and the Greenbush. These are recognizable but do not in general carry important coal deposits in Illinois, though they may elsewhere. Above the Greenbush are the Liverpool, with its No. 2 coal; the Summum, with No. 4; the St. David, with No. 5; the Brereton with No. 6; the Sparland with No. 7; the Gimlet with no coal, at least in northern Illinois where this section was made, and above this the Trivoli with No. 8 coal.

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[2] H. R. Wanless and J. M. Weller: *Ibid*, 1005.

These coal beds are well known in the part of the state indicated. Several of them are definitely placed over a large part of Illinois and a few in Indiana, though lack of satisfactory geological methods has led to a few cases of mistaken identification.

Correlation of Western Interior Coals with Those of Illinois

Important as is this work in Illinois, it becomes more important and more interesting if some at least of the coals of that state can be traced into the southwest. Messrs. Wanless and Weller believe that they have been able to do this, though they speak with less certainty of the coals of Iowa and Missouri than of those of Illinois, and their work has been limited to Iowa and northern Missouri, and there has not been opportunity for the detailed work that has been done in Illinois. Nevertheless, as they say, "the succession of beds in the lower part of the Pennsylvanian system in most basins is so generally similar that it is possible to make some definite correlations."

In southeastern Iowa, where all but the lowest beds are absent, they believe that they have recognized, near the top of the Pennsylvanian in that district and near the surface, the No. 2 coal of Illinois, which occurs in the Liverpool cyclothem of the latter state.

In Davis and Appanoose counties, in Iowa, on the south line of the state and a little east of the center, which are important in coal production, the lower part of the stratigraphic column has not been so well correlated as it has farther to the east and south but it extends much higher. Here they believe that they have identified the No. 2 coal of the Illinois Liverpool cyclothem and, what is probably more important in working to the southwest, they identified the Mystic coal with the No. 6 of Illinois.

Of the two counties mentioned, Appanoose is farthest to the west; northwest of Appanoose is Lucas County and here many of the beds found in Davis and Appanoose counties are also found. Tentative correlations indicate that the Mystic coal is absent, that a seam mined along Swede Hollow is coal No. 5, that the coal locally known as the Wheeler is No. 4 (we shall find this coal again much farther to the southwest), and that a coal known as the Whitebreast is the No. 2 (and this also can be traced to the southwest though not as far as No. 4).

The upper coal that is mined in the northeastern corner of the county near drainage level is probably the Seville coal, and the "lower" coal 40 to 60 ft. below is possibly in the Babylon cyclothem or even lower stratigraphically.

Passing to the southwest corner of Iowa, a district in which Wanless and Weller did not work, we find a coal known as the Nyman, which is known to cross northwestern Missouri and southeastern Nebraska and to occur as far south as what is known as the Osage district, near the middle of Kansas north and south. It is a thin coal in Iowa and too thin to be workable in Kansas on a commercial scale, and lies stratigraphically far above the Pennsylvanian strata present in western Illinois, which extend about to the top of the Illinois equivalent of the Des Moines series.

In Missouri the Liverpool cyclothem has apparently been distinguished in Boone and Randolph counties, which are near the center of the state east and west and north of the Missouri River. Here the Bevier coal, using the Missouri terminology, is mined and this, if the identification is correct, as it probably is, is equivalent to the No. 2 of Illinois.

Farther west, at Lexington, in Lafayette County, approximately 40 miles east of Kansas City, coal is mined known as the Lexington. Probably it is the No. 6 of Illinois, or it may be that the No. 6 is not developed at this point. The lower strata here are made up of the St. David, Summum and Liverpool cyclothems and bear the Summit, Mulky and Bevier coals.

Along the border between Missouri and Kansas, in the neighborhood of Fort Scott, approximately 90 miles south of Kansas City, there is a district in which some mining has been done and here the strata resemble the standard Illinois section.

The Williams coal of Missouri (a local name) is the Bevier, which is the Whitebreast of Iowa, and probably the No. 2 of Illinois. The cyclothem that has been identified with the Summum includes a coal that has been recognized over a wide territory. It is known as the Lower Fort Scott of Kansas, the Mulky or Macon City of Missouri, the Wheeler of Iowa, and is traceable a considerable distance into Oklahoma. It extends over a very wide territory but is nowhere a thick coal. A coal locally mined as the Butler is possibly the No. 7.

In the present state of the published knowledge of geology, it is unwise to attempt to go much beyond the Oklahoma border in identification. In a later portion of the paper, which deals with the Oklahoma coal, a few possible identifications will be mentioned. But in this place it seems desirable to speak positively only of the Fort Scott coal, and this can be traced definitely to the Arkansas River.

The present state of the work is illustrated in Fig. 1. This shows probable correlations, but it is not safe to say that all of them are certain. For example, the bed known as the Cherokee in Oklahoma may not be the same as the Lower Weir-Pittsburg of Kansas and Missouri.

It should be said concerning this method of identification of beds that it requires a large amount of detailed work. It is known that the cycles become thicker or thinner from place to place and that some beds are missing. Missing beds can be accounted for principally in two ways: (1) by erosion and (2) by failure of the oscillation to extend deep enough to permit the deposition of marine limestone, or to extend high enough to permit the deposition of sandstone.

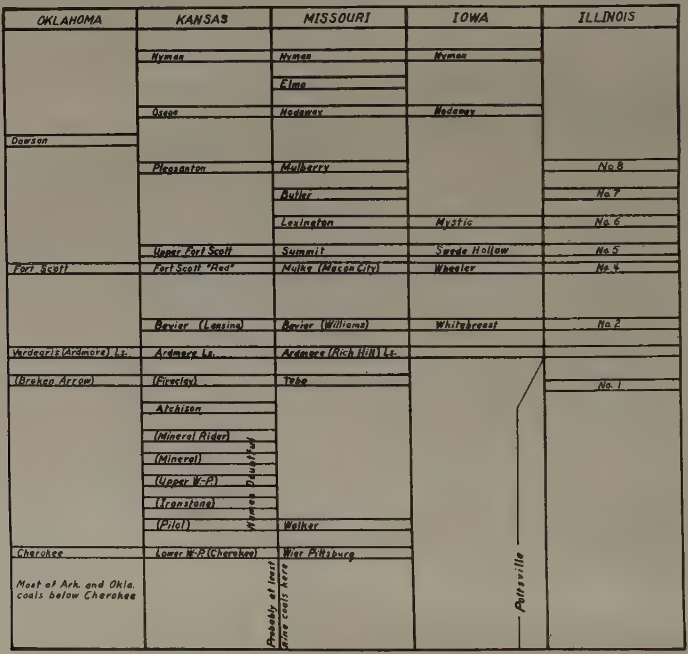

FIG. 1.

The work is being pushed forward by a number of geologists and when their information is correlated we may expect positive identification of at least some of the coals over much wider territory than has heretofore seemed possible. At the present time, and for several months in the past, geologists of the United States Geological Survey have been at work in eastern Oklahoma and Arkansas and when their work is published some of the confusion that attends the identification of beds in those districts will probably be removed.

Coal Fields of the Western Interior Province

For detailed knowledge, the reports of the various state geological surveys, of the U.S. Geological Survey, and of the U.S. Bureau of Mines, should be consulted. The writer has attempted here only to bring together a few of the most important and interesting of the many details that have been collected and published. In general direct quotations have not been made, in spite of the fact that most of the following material has been drawn from these various publications.

Iowa

In general, the lower coals of Iowa are very irregular, thickening and thinning rapidly from point to point, sometimes disappearing except for a thin streak or perhaps altogether, and being replaced by another bed, which may occur a little above or a little below the first. In general these beds, in their workable portions, have a thickness averaging about 4 ft., though in places it may run up to 7 ft. So rapidly do some of them thin out that seldom do more than 300 or 400 acres of workable coal occur together.

The location of mining industries has been determined largely by topography, the first workings being where the various coals were exposed in river valleys and extending only gradually backwards into the uplands. The irregularity of the beds and the location of the workings together will account for the impossibility of making any very accurate estimate of resources.

According to M. R. Campbell, the original supply of the state was 29,160,000,000 tons. In the opinion of George S. Rice, less than 25 per cent of the carboniferous area, except in that known as the Centerville district, where the Mystic coal is mined, contains workable coal.

With the exception of the Mystic and the beds above it, those most worked lie generally within 50 ft. of the Mississippian limestone.

This Mystic coal is an exception to the inconstant character of the lower beds. As we have seen, it extends far to the southwest into Missouri, where it is mined at many places and certainly goes as far as about 40 miles east of Kansas City in this part of the state. Apparently when this seam was deposited, conditions had become favorable, both with regard to plant growth and sedimentation, over a much wider area than previously. It may be possible that erosion and sedimentation had so far leveled off the surface, which was alternately land and sea, that a slight depression extended swamp conditions over a very wide area.

In Iowa, this bed extends over an area of at least 275,000 acres and the coal is present everywhere, except where it has been removed by old erosion channels. Apparently it formerly extended an unknown distance to the north and east of its present line of outcrop.

Also, the coal is remarkable in the uniformity of its characteristics. In Iowa it does not vary more than a few inches either way from a thickness of 30 in., and always carries in its middle portion a clay parting that is from ½ to 2 in. thick. Concerning this parting, one has to wonder whether it is our old friend the "blue band" of Illinois No. 6, and can hardly doubt that it is. It seems easier to understand the immense area of deposition of this bed than to account for the conditions that made this streak of impurity apparently as widespread as the bed itself. It is certainly an interesting subject for study. In northern Missouri the bed has two partings but at Lexington, from which it takes its name, it has only one. This coal is in the Labette shale.

In the southwest corner of Iowa two coal beds are found within workable distance of the surface. The lowest, called the Nodaway in Iowa and Missouri and known as the Osage in Kansas, is near the top of the Wabaunsee formation. It is said to cover a larger area than any other in Iowa, but it is thin, running from about 6 in. to perhaps 3 ft. and averaging only about 16 in. Because it is widespread in both Iowa and Missouri, it is worked at numerous places in spite of its thinness, and it forms the only workable bed of one of the coal fields of Kansas. This coal occurs in the Severy shale which, until recently, was placed in the Shawnee formation but has now been assigned by the Kansas Geological Survey to the Wabaunsee formation.

One coal, not noticed in Iowa and apparently not in Kansas, appears above this Nodaway coal in Missouri, where it is known as the Elmo. Apparently this is worked only in Missouri and not on a large scale.

Near the top of the Wabaunsee is a thin coal known as the Nyman in southwest Iowa, southeast Nebraska and northwest Missouri. It extends also into Kansas but apparently is not of workable thickness. It seems to have been worked in Iowa only for local consumption. Incidentally we may note that apparently this is the only coal that has ever been worked in Nebraska, and that working was on a very small scale and has been discontinued.

Though this coal is not often workable, it is well worth attention because it occurs within a short distance of the top of the Wabaunsee, which, according to the latest investigations, must be considered the topmost group in the Pennsylvanian. After this comes the Permian with only the slightest of unconformities here and with conditions that seemed to continue those of the Pennsylvanian but that did not produce coal in spite of that continuance. We know, of course, that the retreat of this Mid-Continental arm of the sea was a gradual thing and that the marine conditions continued longer in the southwest than in the northeast, and there may be some higher coals in Oklahoma. However, this seems doubtful because the sea apparently was growing more salty and it is doubtful whether any later condition was favorable to coal formation.

There is, however, a streak of coal a little higher up and probably not more than 15 ft. from the top of the Pennsylvanian.

QUALITY OF COAL IN IOWA

Bed	Proximate Analysis, Per Cent Moisture	Volatile Matter	Fixed Carbon	Ash	Heat Value,[a] B.T.U.
Mystic	17.1	35.4	40.4	7.1	10,931
		(Moisture and ash free)			14,422

[a] U.S. Bur. Mines *Tech. Paper* 269. Only one analysis is quoted, in part because of the difficulty of determining the beds represented by the many analyses given in the *Technical Paper* and in part because many of the determinations of heat value were not made with an oxygen bomb.

Missouri

As is shown by the maps, rocks of Pennsylvanian age cover a large territory in northern and western Missouri. They are excluded from the southeastern part by the Ozark uplift, against which the edges of succeeding strata were deposited. In general the strike of these beds is from northeast to southwest. More particularly, it may be said that north of the Missouri River and west of the bend where this river turns south, approximately one-third of the way across the state from west to east, the Pennsylvanian rocks dip nearly due west about 8 ft. per mile, but east of that general locality they are nearly level. In the central part of the state, south of the Missouri River, the general dip is about 6 ft. per mile to the west, but in the southwestern part, where the Lower Weir-Pittsburg bed is mined, the strata have the same dip as in southeast Kansas, which is about 25 ft. to the mile in a direction about 70° west of north.

The principal coal beds are tabulated as follows in U.S. Bureau of Mines *Technical Paper* 366 on Analyses of Missouri Coals:

Wabaunsee formation:
  Nyman (Atchison County)
Shawnee formation:
  Elmo (Nodaway County)
  Nodaway (Nodaway County)
Pleasanton formation:
  Ovid (Sullivan County)
  Mulberry, Foster (Bates County)

Cherokee shale:
  Lexington (Mystic) (Fort Scott red coal), Mendota
  Summit
  Mulky, Macon City
  Bedford, Upper Rich Hill
  Weir-Pittsburg upper
  Bevier, Lower Rich Hill, Weir-Pittsburg Lower
  Tebo
  Brushy Hill
  Jordan (Henry, St. Clair, Cedar, Dade, and Barton counties)
  Monteserrat (Johnson and Lafayette counties)
  Cainesville (Harrison and Mercer Counties)
  Eureka (Macon and Linn counties); Waverly (Lafayette county)

It should be said, however, that stratigraphic work done since this list was published makes some correction necessary. The Lexington is not the Fort Scott, the Bevier is not the same as the Lower Rich Hill and the Weir-Pittsburg lower, and it is doubtful whether any one bed can be called a "Weir-Pittsburg upper." The Bevier occurs just above the Ardmore limestone and, in the western part of Missouri, is about 100 ft. below the top of the Cherokee, while the Weir-Pittsburg lower is about 250 ft. below the top of the Cherokee.

Conditions favorable for widespread and uniform coal formation appear to have occurred earlier here than in Iowa, for though the lower beds are not persistent the Tebo is persistent and is present almost everywhere northwest of its line of outcrop. In point of production in Missouri, it is second only to the Bevier.

The table from *Tech. Paper* 366 does not by any means name all of the coals and traces of coals that have been found, but only those that are of commercial importance at some place. Thin local coal beds occur in the Wabaunsee, Shawnee, Douglas, Pleasanton and Henrietta formations, and other coals may be present in the lower part of the Cherokee shale near the Kansas State line.

The four continuous workable beds, from below upward, are the Tebo—which cannot be identified with any other bed in this part of the province but which perhaps may not be far from the lower Ardmore of Oklahoma—the Bevier, the Mulky (the Fort Scott of Kansas) and the Lexington, which is the Mystic of Iowa.

Considering only these four beds, the thicknesses, although variable, are somewhat as follows: The Lexington bed averages 18 to 23 in. in thickness and the Mulky about 20 in. The Bevier averages about 4 ft., though in places it increases to 5½ to 6 ft. It has a clay parting 8 to 12 in. above the bottom. The Tebo varies somewhat in thickness and its maximum seems to be about 24 to 30 inches.

It is estimated that the original coal supply of the state amounted to 79,000,000,000 tons. Mining has not exhausted any large amount of this

QUALITY OF COAL IN MISSOURI

Bed	Proximate Analysis, Per Cent Moisture	Volatile Matter	Fixed Carbon	Ash	Heat Value,[a] B.T.U.
Mulberry	9.2	32.1	44.2	14.5	11,210
		(Moisture and ash free)			14,690
Lexington	14.9	31.3	41.0	12.8	10,370
		(Moisture and ash free)			14,360
Mulky	8.7	39.9	42.9	8.5	12,070
		(Moisture and ash free)			14,590
Bevier	13.3	33.9	42.0	10.8	10,810
		(Moisture and ash free)			14,250
Tebo	14.9	36.4	41.3	7.4	11,120
		(Moisture and ash free)			14,320

[a] U.S. Bur. Mines *Tech. Paper* 366.

coal, except where stripping is employed. The area of the Lower Weir-Pittsburg suitable for stripping has been reduced almost to zero and a large amount of coal has been taken from the outcrop of the Mulberry, stripped near the western border of the state about 50 miles south of Kansas City.

Kansas

While strata of Pennsylvanian age cover a large portion of eastern Kansas, the part certainly known to be underlain by workable coal is limited. The state has three principal districts; namely, Southeast Kansas, Northeast Kansas, and the Osage City field, which is a comparatively small district lying about 30 miles south of Topeka.

The principal coal-bearing formation is the Cherokee, and in this the bed most worked is the Lower Weir-Pittsburg. This bed occupies a restricted area in Crawford and Cherokee counties in the extreme southeast corner of the state and projects a short distance into Missouri. On the east it is limited by the outcrop, and on the west it has been cut off by an old erosion. As we have no means at present of knowing whether this erosion was caused by a stream or by a large body of water, it is impossible to say whether the coal can be picked up again to the north and west.

Some drilling has been done in an effort to extend the field, but this drilling was not carried far enough to cross the valley, which might have been excavated by a river of any considerable size. Also the holes were not deep enough to allow for any serious displacement of the bed by increase of dip or by faulting. Holes farther away have all been drilled with churn drills in the search for oil, and their records in regard to coal are practically worthless. For these reasons, we must say that there is no proof of either the presence or absence of this bed beyond the line where it is cut off by erosion from 6 to 12 miles northwest of the outcrop.

In northeastern Kansas is an area of considerable size where coal may or may not be present. Mining has been carried on principally at Leavenworth and Lansing. The bed mined is the Bevier of Missouri and the thickness of the coal averages about 19 in., though it is commonly called a 22-in. bed. About 27 ft. below this is another bed of approximately the same thickness, which has never been worked. Without any knowledge of this coal beyond its relation to the Bevier, it cannot be identified, but one may risk the guess that it is the Tebo. Numerous thin coals are encountered in this district but apparently only those mentioned can be considered workable.

At Atchison, near the northeast corner of the state, a coal some 325 ft. below this Bevier bed which is worked at Lansing, was opened and operated for a few years. Coal was discovered by drilling and another bed 28 in. thick was penetrated 64 ft. below the one just mentioned. At Atchison the Bevier bed was passed through and found to be 22 in. thick. A prospect hole in one of the Leavenworth mines showed coal about 3 ft. thick at the horizon of the Atchison coal. This does not prove the continuous existence of this coal over the 20 miles between Atchison and Leavenworth but it strongly suggests it. Moreover, this coal has been encountered in other drill holes and all indications are that there is a fairly large area of workable coal but there is no positive knowledge whatever on the subject. The extreme depth of about 1000 ft. at the only shaft where this coal has been worked makes it improbable that it will be utilized in the near future.

It has been a common statement that the coals of eastern Kansas, Missouri and Iowa were marginal deposits and that no great width of beds could be expected. Unquestionably they are marginal coals, and the Cherokee shales, for example, are not coal-bearing farther to the west where they become important in production of oil and gas. The Pennsylvanian as a whole is not coal-bearing where it appears again far to the west, but no one knows how far the coal beds extend from the eastern outcrop. But marginal need not imply a limit of 6 to 12 miles, as it seems to have done when it was applied to the Weir-Pittsburg bed of southeast Kansas, where the western margin is quite readily defined. The Bevier coal mined at Lansing, and formerly at Leavenworth, occurs from 60 to 70 miles, measured along the dip, from the outcrop. Occurrences of coal at this distance from the outcrop indicate that, if these are to be considered marginal deposits there is little reason to consider them merely narrow belts. Moreover, there is nothing to indicate that the operations at Lansing and Leavenworth are anywhere near the western border of the field. On the other hand, there seems good reason to believe that large areas of which we now know nothing may contain the coal of at least this Bevier bed. In fact, the writer is somewhat inclined to compare it in area covered with the Pittsburgh bed of western Pennsylvania, Ohio and West Virginia.

In the southern part of Kansas the Cherokee produces oil, somewhat west of the coal fields, and it is probable that the coals really are marginal deposits and that on passing to the west the water became too deep for swamp vegetation but that an abundant water life furnished the organic material from which oil and gas were produced, instead of coal. But the belt of possible coal is not narrow. Whether the limit of coal to the west in Northern Kansas is as narrow as in southern Kansas may be doubted, since we have no oil and gas production to give a fairly definite indication of the change of conditions.

According to F. C. Greene, of the Missouri Bureau of Geology and Mines, the stratigraphy of the district in northeast Kansas and northwest Missouri is sufficiently different from that of southeast Kansas and the bulk of Missouri to indicate the possibility that this is a separate coal field.

The third district in Kansas, commonly called the Osage, lies about 30 miles south of Topeka. The coal here occurs in the Severy shale, considerably above any other productive beds of Kansas, Missouri or Iowa, except the Nyman and the Nodaway. It is just above the Topeka limestone, which is now considered the top formation of the Shawnee and, therefore, lies at the base of the Wabaunsee. It is almost certainly the Nodaway coal of Missouri and southwest Iowa. This district in Kansas never contained a large tonnage of coal. At one time the writer estimated the original tonnage at about 89,000,000 tons. A considerable portion of the workable part has been removed.

About 325 ft. stratigraphically above the Osage coal is the horizon of the Nyman coal in Kansas. We have seen that this has been produced in southwest Iowa and that there was once a little working in southeast Nebraska. Probably a little gophering has been done along the outcrop in some places in Kansas but there is no reason to believe that this bed reaches commercial proportions at any place.

It is known that there are various coals below the lower Weir-Pittsburg in the southeast Kansas district, but none are being worked. Coals below this horizon furnish most of the tonnage produced from Oklahoma and Arkansas.

Until recently it was believed that there was only one bed of any considerable extent, known as the Lightning Creek, or Upper Weir-Pittsburg, overlying this Weir-Pittsburg coal, but it was believed that there might be one or two others occupying small areas. Recent work, not yet published, indicates probably seven beds, some of which can be correlated with Missouri beds, but does not indicate any very large area of productive thickness.

A well known coal, though one that has never been produced extensively in Kansas, is the Lower Fort Scott, which has been identified with the Mulky of Missouri, the Wheeler of Iowa and the No. 4 of Illinois. In other words, it is one of the most widespread coals west of the Mississippi. It occurs near the top of the Cherokee, just under the Fort Scott limestone, which is the lowest member of the Marmaton group. It is only from 15 to 20 in. thick, but a considerable tonnage has been produced by stripping. Some of the earliest coal produced in Kansas was from this bed, where workings were undertaken before the development of the thicker coal of the southeast district.

Approximately 10 ft. above this is a bed known as the Upper Fort Scott in Kansas, as the Summit in Missouri and the Springfield No. 5 coal of Illinois. It has been worked to some extent in Missouri, but very little coal has been produced from it in Kansas.

The Lexington coal of Missouri seems not to be developed in workable thickness in this portion of Kansas. Above the Cherokee, in the Bandera shale, just above the Pawnee limestone, is the Marmaton group, which contains the coal mined in the vicinity of Pleasanton in Linn County, known in Missouri as the Mulberry coal.

The Kansas City group, just above the Marmaton, contains a coal known as the Thayer, mined in a small way along the outcrop.

The Douglass group contains a thin coal in the Lawrence shale, which has long been mined in a very small way but recently has given employment to many workers along the outcrop. The output is not large but the coal is rather widely distributed by truck.

There seems to be no other workable coal until the Osage is reached. It is a most interesting fact that here, near the western border of the Interior Province, there is a coal that cannot be very far in age from the Pittsburg bed of western Pennsylvania and eastern Ohio.

For the reasons given above, it is impossible to make any reliable estimate of the coal resources of the state. Apparently only one bed, the Lower Weir-Pittsburg, has ever been explored for extent. The known area of this bed was easily determined, and it was estimated that the original content was about 295,000,000 tons. The parts of this bed suitable for stripping are approaching exhaustion, but there is a considerable amount of coal left that can be reached by underground mining when this again becomes profitable. Over the rest of the state there is so little known of the extent of the workable beds that one cannot afford to risk an estimate.

QUALITY OF COAL IN KANSAS

	Proximate Analysis, Per Cent				
Coal	Moisture	Volatile Matter	Fixed Carbon	Ash	Heat Value,[a] B.t.u.
Cherokee (Lower Weir-Pittsburg).	6.6	33.4	50.4	9.6	12,540
		(Moisture and ash free)			14,970
Leavenworth (Bevier)......	12.0	35.2	39.1	13.7	10,720
		(Moisture and ash free)			14,420
Pleasanton (Mulberry).....	8.2	31.6	45.4	14.8	11,420
		(Moisture and ash free)			14,830
Osage[b]....................	7.29	36.62	45.78	10.31	10,630
					12,900

[a] U.S. Bur. Mines *Tech. Paper* 455.

[b] Young and Allen: Univ. Kansas Experiment Station *Bull.* 13, 191.

Oklahoma

When we try to extend into Oklahoma and Arkansas, the names, the locations in the geologic column, and the qualities of the coals with which we are familiar in Iowa, Missouri and Kansas, we encounter such difficulty that at present the task seems impossible. This is due in part to change of conditions and in part to lack of knowledge. To a large extent lack of knowledge is due to the change in conditions, which become much more difficult to interpret. An intensive study of the coal fields of southeast Kansas, Oklahoma and Arkansas, has been carried on for some time by the United States Geological Survey, but unfortunately many of the results of this work are not yet available.

In Iowa, Missouri and Kansas, there is a series of Pennsylvanian strata, varying enough in thickness to make correlations doubtful in cases but not departing far from the horizontal. In the northern part of the district they are covered with glacial drift and for that reason study has been somewhat difficult, except in the southeast corner of Iowa. But, on the whole, it is possible to find exposed an orderly sequence of rocks running from the Mississippian to the top of the Pennsylvanian, dipping gently to the west, or north of west, and outcropping in a belt of varying width from eastern Iowa toward the southwest. These conditions continue into the northeast part of Oklahoma, but many correlations cannot be extended safely much beyond the northern border of the state because there has not been sufficient detailed study. In the east-central and southeast portions of the state conditions are much more complicated. In the first place, the thickness of the beds increases tremendously, and this is true even in the northern portion of the state. For example, the Cherokee alone increases from about 450 ft. at the north line of the state to 1000 ft. in a distance of about 50 miles—in that undisturbed portion of the state where correlation is easiest. At least two coal beds can be traced into it but it is far from being as important in production as the portion of the state south of the Arkansas River. There conditions are seriously changed. The coal area extends from the Arkansas River south to the Arbuckle and Ouachita Mountains. As a continuous coal basin, it extends from the eastern tip of the Arkansas fields to the limits of known beds in Oklahoma, some 80 miles west of the Oklahoma line, although this area is by no means entirely coal-bearing.

The thickening of the beds as they pass into Oklahoma has been mentioned but without indicating the full extent of this thickening. One illustration is enough; the Cherokee shales, which have a total thickness of about 450 ft. at the Kansas line, have reached a thickness of about 9000 ft. at the southwest end of the coal fields. Perhaps it is more accurate to say the beds that represent the same geologic interval as the Cherokee shale.

Another complication is the fact that the bottom of the Cherokee, as we have found it in the northeast portion of the Province, is not the bottom of the Pennsylvanian as we move into Oklahoma. Not only is there a great thickening of beds but there is overlap, or at least it is believed that both processes must be taken into account in the study of the stratigraphy of this district.

The nomenclature in use is different as well as the thickness of beds. That which in northeast Oklahoma is called the Cherokee is known in south-central and eastern Oklahoma as follows:

Senora formation		Cherokee formation
Stuart shale and Thurman sandstone		
Boggy shale		
Savanna sandstone		
McAlester shale	Winslow formation with Bluejacket sandstone at base	
Hartshorne sandstone		
Atoka formation		

The Atoka carries some coal in Arkansas but none has been produced from it in Oklahoma. Immediately above this is the Hartshorne sandstone, which is one of the principal markers of the coal field. Just above it are two of the most important beds mined, known as the Lower and Upper Hartshorne and, in a large part of the field, these are the most valuable coals. The McAlester coal is some 1200 to 1500 ft. above the Hartshorne coal. In fact, this entire series of strata is coal-bearing. The Savanna sandstone, above the McAlester, contains at least three workable beds and above this is the Boggy with two. Above these is the Stuart, which apparently contains none, and then the Senora, with one.

On the whole, the conditions of eastern Oklahoma and western Arkansas are too complicated for discussion in a general paper. Detailed discussions are given in the publications of the State Geological Surveys, and in papers under preparation by the United States Geological Survey.

Resources of Oklahoma.—The area actually underlain by coal is approximately 12,000 square miles. While numerous beds can be found, only seven are of economic importance. Workable beds range in thickness from 2 to 6 ft. and in a few localities reach 7 to 8 ft. The average thickness is about 4 ft. The Oklahoma coals are thicker on the average than those to the northeast.

With regard to correlations, it is not certain whether the Lower Weir-Pittsburg bed extends into Oklahoma, but it is entirely probable that it does. Drilling to the west of the southern end of this bed in Kansas has shown coal at the depth at which this bed should be found, and the same statement can be made of drilling across the border in Oklahoma. The depth, however, is considerable, running in the neighborhood of 600 or 700 ft., and it is doubtful whether mining can be carried on profitably under present conditions. Drilling in the northern part of Oklahoma indicated that the coal was irregular and that it would be difficult to locate workable areas of more than 1200 to 2000 acres.

The Fort Scott coal does extend well into the state. Thus it is possible to trace this bed, apparently continuously, from the neighborhood of the Arkansas River through northeast Oklahoma, southeast Kansas, Missouri and Iowa, and probably to correlate it with the No. 4 coal of

Illinois. In Oklahoma, as in Kansas and Missouri, the coal is not thick and is not largely worked.

Somewhat lower than this, and from 50 to 75 ft. below the top of the Cherokee shale, is a coal known as the Cherokee, which has been mined to a considerable extent in the northeast portion of the state. Frequently this has been called the same as the lower Weir-Pittsburg of Kansas, but a definite correlation does not seem warranted at present. The highest coal, stratigraphically, of commercial importance is the Dawson, which has been mined along a considerable extent of outcrop by stripping. It has not yet been correlated with other beds.

The coal known as the Cherokee ranges in thickness from 10 to 48 in., and averages from 18 to 22. The Fort Scott varies from a thin seam to a maximum of 21 in. The Dawson is from 20 to 30 in. thick.

QUALITY OF COAL IN OKLAHOMA

	Proximate Analysis, Per Cent				
Coal	Moisture	Volatile Matter	Fixed Carbon	Ash	Heat Value,[a] B.T.U.
Dawson....................	6.7	36.3	48.3	8.7	12,540
		(Moisture and ash free)			14,820
Hartshorne (low volatile)...	3.2	17.3	76.3	3.2	14,580
		(Moisture and ash free)			15,580
McAlester.................	3.5	36.4	53.3	6.8	13,360
		(Moisture and ash free)			14,890

[a] U.S. Bur. Mines *Tech. Paper* 411.

Arkansas

The disturbed geological conditions of eastern Oklahoma extend into Arkansas but become so far intensified that the coal decreases in volatile matter and approaches nearer and nearer to a true semi-anthracite.

Those who are familiar with geology will remember that the Ozark uplift was already in existence at the time of the deposition of the Pennsylvanian rocks. To the south and east was an old land from which a great thickness of sediment was washed into the sea, which separated it from the Ozarks. This mass of sediment was compressed and folded and faulted into the Ouachita Mountains. The folding is severe here, as it is in eastern Oklahoma, where dips of 60° to 80° are not uncommon. Naturally, in a region of so much disturbance erosion has been very prominent and the workable fields left occupy synclines left by erosion of the intervening anticlines. The following facts regarding the field are quoted from Branner, State Geologist of Arkansas[3]:

> The coal lands of western Arkansas are concentrated in a rather narrow belt along the western edge of the regional syncline between the Ozark and Ouachita mountain regions. The Arkansas field is an eastern extension of the Oklahoma field . . . The productive coal measure beds are distributed principally along the major synclinal trough which extends eastward from Fort Smith through Crawford, Sebastian,

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[3] G. C. Branner: Outline of Arkansas' Mineral Resources, 109–111.

> Franklin, Johnson, and Logan counties to the vicinity of Russellville, Pope County, a distance of about seventy-five miles. The area is about twenty miles wide and decreases in width toward the eastern end. The remainder of the developed coal field extends south of Fort Smith into southern Sebastian and northern Scott counties. There are other small areas in the Arkansas River Valley region north of the Ouachita Mountains but these are commercially unimportant.
>
> The area of workable coal lands within the State is estimated at between 300 and 350 square miles and the total area covered by coal measure beds at about 1,620 square miles.
>
> The coal-bearing formations of Arkansas are confined to the lower part of the Pennsylvanian series of beds which in Oklahoma is usually classified as the McAlester shale. (This is a portion at least of what farther north is known as the Cherokee shale). In Arkansas, this McAlester shale has been divided into three parts as shown by the following table:

COAL-BEARING ROCKS OF ARKANSAS[a]	THICKNESS, FT.
Savanna formation	200 +
McAlester group—	
Paris shale	600
Fort Smith formation	400–500
Spadra shale	400–500
Hartshorne sandstone	50–400
Atoka formation	1,500–5,000

[a] A. J. Collier: The Arkansas Coal Field. U.S. Geol. Survey *Bull.* 326.

> As in Oklahoma, by far the larger coal production of Arkansas has come from the top of the Hartshorne sandstone and is known as Hartshorne coal. . . . The upper portion of the Fort Smith formation has been mined for coal near Charleston, Franklin County, and produces 'Charleston' coal, and the middle of the Paris formation is mined near Paris, Logan county, and produces 'Paris' coal.
>
> This coal field occupies the lowest portion of the regional syncline between the Ouachita and Ozark regions and 23 major anticlines have been mapped in the coal-bearing area.
>
> The coal increases in devolatilization in a fairly consistent manner from the western state line eastward. It contains from three to six times as much fixed carbon as volatile combustible matter in the western portion and from seven to nine times as much in the eastern. It thus grades from bituminous through semi-bituminous to a semi-anthracite. About 15 per cent of the coal produced is semi-anthracite.

As shown by the statement just quoted, the coal fields of Arkansas are very narrowly limited in resources. The character of the coal is such as to give it principally a domestic market where it is favored because of its low content of volatile matter.

QUALITY OF COAL IN ARKANSAS

BED	PROXIMATE ANALYSIS, PER CENT				HEAT VALUE,[a] B.T.U.
	MOISTURE	VOLATILE MATTER	FIXED CARBON	ASH	
Hartshorne	3.2	14.8	72.7	9.3	13,590
		(Moisture and ash free)			15,530
Paris mine	2.8	14.7	73.4	9.1	13,770
		(Moisture and ash free)			15,620

[a] U.S. Bur. Mines *Tech. Paper* 416.

RESOURCES OF THE PROVINCE

The bulk of the coal resources of the Western Interior Province has hardly been touched. All of the operations in this field are rather shallow and hardly any attempt is being made at deep mining.

In the northeast corner of Kansas mining has been carried on at greater depths. The shaft at Atchison produced coal from about 1000 ft. At Leavenworth operations were carried on at about 720 ft., but workings at both of these places have been abandoned. The only mine now operating at any great depth is the state mine, worked by prison labor at Lansing, Kansas, where the Bevier coal is mined at about 720 ft. There is abundant evidence of the occurrence of coal at anything from this depth to the outcrop, some of it near large markets. For example, it is known that there is coal at a depth of about 500 ft. in the vicinity of Kansas City—probably on all sides of the city.

The mining of these comparatively thin coals at considerable depths is not profitable at the present time and is not likely to be as long as oil and gas are abundant and the cheaper outcrop coals can be mined and coals are brought in from the Illinois field, where they can be produced cheaply enough to compete with the coals of the Missouri valley.

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